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Nineteen Hundred and Eleven.

In any valuation of business prospects in this country one fact must be ever kept before the mind's eye. This is that America is a youthful nation with the faults and virtues of youth. Americans are as different from Chinese as if the latter lived in Saturn. They live in the past and worship their ancestors. We live in the future and look for a continuation of our individual spiritual entities after death.

Now the foremost characteristic of youth is the profound major chord of optimism dominating its entire life and the resultant reaction from that attitude for short intervals. It is thus true that in our national life we have periods when fear and doubt challenge the attention. We have seen alternations in our industrial life as in the life of a youth ever since the Constitution was adopted in 1789 and Benjamin Franklin made his now famous remark that the picture over the presiding officer's chair represented "the rising sun, not the setting sun." Summarized briefly, we have seen recently the depression of 1893-6, the boom of 1900-02, the rich man's panic of 1903-04, the flood tide prosperity of 1905-06 and the widespread panic of 1907 with its subsequent period of retrenchment.

Furthermore, we have observed the hysteria of 1907 vanish in eighteen months and the national spirit reverse itself in the time of elation and apparent prosperity from July, 1909, to September, 1910. This was exaggerated in point of fact and there has been manifested a general mental attitude of doubt and perplexity in the past sixteen months, gradually turning into a time such as the present, when confidence and assurance begin to outcrop. In a highly exaggerated form, alienists recognize a similar course of mental events in what they denominate as "alternating insanity." Here, of course, the unintelligence is most pronounced. In the milder neurasthenic cases there are the same phenomena. If we assume the Hegelian theory of a national individual soul which is at least a good working hypothesis we can compare these alternations in business to the alternations that exist in any youthful mind. Now, in sane individuals these alternations are mild. An even and balanced mind discriminates between unreasoning hope and an abiding faith, between hysterical fear and sensible caution. It can be laid down as a fact that our panics have been growing less severe in the past fifty years. We perceive a greater extension of business intelligence throughout the United States and we believe that our industrial organism is animated by the general diffusion of the laws of economics and sound finance. It is a direct and logical deduction to say that these are related and to compare our now less pronounced industrial alternations to those of a heavily dampened pendulum. This is to our way of thinking a salient feature of our modern business life. In short, we are growing older and more sensible.

Unquestionably the rise and fall are at any rate less secure. There is a tide in the mental affairs of men which receding leaves bare a few unsightly feats. But always as the Saxon poet says "Bac comes flowende flood aefter ebban."

So much for the general and philosophical argument of the situation. The specific events, we believe, give evidence of a marked agreement between theory and practice. They all seem to point to a general sane expansion and the removal of fear. The most striking event of the past year was, of course, the two decisions of the Supreme Court in the two "Trust" cases. This was most difficult, but was accomplished with judicial dignity, perspicacity, and practical sense. "The rule of reason" simply means that in interpreting any statute common sense takes cognizance of notorious social conditions and that "de minimis lex non curat" is not to be perverted by a meticulous consideration of verbal minutiæ. It means, too, Americans only allow the existence of organizations of labor or of capital provided their actions are governed by intelligence. The application of the Sherman law to labor unions is easy and just. It is of special significance and is directly applicable to the concatenation of events that lead up to the "McNamara case."

This is to our mind the second event in point of importance in the past year. It has brought to the labor unions and their leaders a realizing sense of their own responsibility. They see now that "as they sow, so must they reap." It is certainly to the credit of the American workingmen that their utterances on the outcome of the McNamara case have been most sane and most fair. But it was not to their credit that they paid no attention to evidences of murder till the murderers confessed.

A third important action in the realms of politics and business was the decision of the Interstate Commerce Commission in refusing to allow the railroads to raise their rates. This was hailed generally by the American people as just and reasonable. It shows we are still a government of the people, by the people, and for the people.

Among events more direct in their relation to business are the immense cotton crop and the fall in the price of the staple to a figure that encourages export trade and domestic buying. Though nine-cent cotton is not an unmixed blessing, it is a compensation to the textile industry.

In addition, our present export trade in all manufactured articles has increased to a degree almost comparable to the expansion of 1898-1900. Indeed our export trade for the past year promises to be in excess of two billion dollars. This has given us large credit foreign balances. We have also seen an all-round increased efficiency in our industrial operations due to the wide diffusion of technical knowledge. Labor, too, while it has not "liquidated" in point of nominal wages—perhaps in view of the higher cost of food products it is not right that it should—has "liquidated" in reality because of increased efficiency and decreased refractoriness.

On the other side of the ledger are the poor crops and the political unrest of the Presidential year, the still prevalent idea that life should be made one round of the "so-called pleasures," the machinations of the ever-present demagogue and of the crass materialistic politician. Both of the latter types we must always expect to have with us as irritants, like the mosquitoes and flies in the warm months.

Therefore, on striking a balance of the debits and credits of the year industrialwise, it is manifest that 1911 has been a very good one for the youthful nation. The year has taught us the worth of the German near-classic proverb, "Arbeit macht das Leben süss, Faulheit stärkt die Glieder." On one side, in the last analysis character and its antecedent hard work are the only thing that counts. On the other side, it is, of course, evident that fat legs do not make character nor is there anything ethical about magnitude.

An American proverb that ought to be classic is "The reason that people who mind their own business succeed is that they have so little competition." This is most pertinent to the American people. At present it is learning the force of the real meaning behind both proverbs. Though a youthful nation we are a thoughtful nation and are increasingly possessing the grace of reason as the years roll by. Business is becoming more sane and more settled. We have not had a bad year in 1911 and there is in sight a probable expansion of business in 1912. It should be expected that this expansion will be likewise more sane and more uniform. Its measure is contingent upon the self-restraint of our people, the size o fthe crops, and upon the release of capital that has been saved in the times of depression.

This release depends largely upon the course of political events. A common-sense President can be neither a reactionary and ultra-conservative nor a shouter for change and still be great—strange as it may appear. The President inspires the prosecuting machinery of the government. He can use this power to alarm or to assure the business world. The President is responsible for the country and a quasi-judicial power must be exercised by his subordinates. But it is quite certain that those who mind their own business cannot be prosecuted for violating the Sherman law and restricting competition.

Concluding this brief review of the past events and of the signs for the future, we can say that barring international complications of a serious nature, the business world should expect a prosperous and happy New Year. A large part will be played in future prosperity by increased efficiency in the metallurgical and chemical industries. To our readers typifying this element in the evolution of this country and of mankind, to all and everyone, this journal extends heartily the compliments of the season.

Copper Metallurgy in 1911.

No very striking developments in the metallurgy of copper have taken place within the past twelve months. Progress has been made along established lines, and such improvements as have been accomplished have, in general, occupied several years in gaining a permanent place. Among these is the general adoption of the basic-lined converter. Although not a new practice, it is only within the past year that there has been a practically complete abandonment of acid-lined converters in favor of those with basic linings. Saving in cost by converting with basic linings has amounted to approximately $1 per ton. A campaign on a single acid lining was rarely of greater duration than a day, with a duty of from 50 to 100 tons of matte; whereas, with the costly magnesite brick lining, 3,000 to 5,000 tons have been treated in a campaign, and the converting cost per ton has been greatly reduced.

Contrary to expectations, the difficulties anticipated in changing the manner of supplying the silica required by the iron of the matte as it became oxidized, proved to be of little importance. While formerly the question of silica supply caused the metallurgist no concern when acid linings were used, because the iron oxide formed would seize the necessary silica from the lining, now with basic linings and silica supplied in loose form, no more concern is felt; for the range of silica to ferrous oxide is so great that fusible slags will form with any of the proportions likely to exist in the converter at any given time. Add to this the fact that converter slags are always resmelted in blast or reverberatory furnaces, and it will be seen that basic converting is very simple and no surprise should be exhibited on seeing successful work being done with basic converters by workmen who, but a few months before, were familiar only with acid practice.

In smelting operations there seems to be no good reason for expecting any marked change in current practice, which has not varied much during the past decade. In this period the giant furnaces were introduced: reverberatories with a hearth area of 116 feet by 19 feet smelting 300 tons of charge a day, with a consumption of as little as 20 per cent coal, and blast furnaces 56 inches by 87 feet at the tuyeres, with a daily capacity of 3,000 tons of charge, using but 8 per cent coke. Such figures force us to the conclusion that there is not much chance for improvement in this direction. In favored localities the application of fuel oil to reverberatory furnaces has been successfully introduced. Efforts aiming at the use of powdered fuel, as in cement practice, have so far met with no success when applied to smelting.

The recent great development of the so-called "porphyries" has attracted much attention to the low-grade copper deposits. With copper present in such small percentages, approximating in many cases but 1.5 per cent, and also very finely disseminated, the question of its beneficiation is really a matter more of ore dressing than of smelting. The great losses during concentration, ranging close to 30 per cent, are the source of much alarm and the cause of much investigation. A study of the subject shows that the losses occur mainly in the very finely divided material, and that the vanners, which are almost exclusively used as fine concentrators, are very poor devices when considered from the standpoint of percentage of saving. Of course, these or similar machines saving sulphides which otherwise would be lost, may be said to be producing a product that deserves to be ranked as "velvet," yet so inefficient are they that attention should be directed to the development of machines which truly merit the name of "slimers."

One large copper company is endeavoring to overcome this slime difficulty by the use of Garfield tables for preliminary dressing, following this with extremely careful classification before final concentration. While this improves the extraction it cannot be said to solve the problem satisfactorily. Two other large companies are attacking the slime problem in another and more logical way, aiming to secure a real slime concentrator. Much promise is given by centrifugal separators, one type of which, operating on material finer than 200-mesh, produces concentrates ranging from 10 per cent to 13 per cent copper with accompanying tailings of less than 0.5 per cent copper, representing a saving of over 83 per cent; this, too, on a rated capacity of 250 tons per day per separator. Active efforts are also being directed toward the recovery of the copper in the tailings of the Lake Superior properties.

We may expect our most important advances in copper treatment along the line of improvement in concentration, or in chemical or electrochemical methods. The latter method of treatment is most promising, especially in view of the fact that electrical power installations are being so generally exploited. Hydroelectrical development in or near many of the mining sections is in rapid progress, and if the resulting power is readily available and cheap, it will have a very important influence in the encouragement of electrometallurgical methods. There are methods of this character, sound in principle, which are only awaiting an opportunity for a demonstration of their merits on a commercial scale and basis. That one or more of the processes recently developed will prove successful seems to be beyond doubt.

As in smelting, so in refining, no striking changes have taken place in the past year. With the exception of most of the Lake Superior copper, all copper is now electrolytically refined. It is interesting to speculate on the future of "Lake" copper, and to wonder how long it will remain a standard; what will be the result of increased depth on the quality; and how soon, because of the presence of a larger percentage of arsenic and other impurities, it will become necessary to electrolytically refine all Lake Superior copper.

A phase of the copper-smelting situation quite as important as the various treatment problems is the attitude of farmers and the United States Department of Justice toward the smelters. In Utah several smelters have been shut down by the courts, some being allowed to operate only under restrictions. In Montana, after winning a suit brought by the farmers to enjoin the operation of the Washoe smelter, and having the United States Circuit Court of Appeals sustain the findings of the Master in Chancery and of the District Judge, the Anaconda Copper Mining Company was threatened with a similar suit brought by the United States Government. Fortunately, after a friendly discussion, it was decided to place the matter in the hands of a commission selected jointly by the government and the company. This agreement is now in effect.

The situation is not so favorable in California, however, and in the near future we may not find an operating smelter in that state. The farmers are very unreasonable, and, flushed with previous victories in debris litigation and over various smelters, seem bent on showing their power. In Shasta county but a single smelter, the Mammoth, is operating, the Balaklala having shut down during the past summer after realizing the futility of attempting to continue under existing conditions.

In Nevada a smelter was threatened with injunction almost as soon as its site was selected, and smelters planned or in course of erection have been warned that injunction suits are certain to follow the blowing-in of furnaces.

Unless friendly agreements are made to submit the question of smoke and fume damage to commissions or to non-partisan boards of arbitration, the operation of copper smelters in many parts of this country will become a practical impossibility.

Gold and Silver During 1911.

The supremacy of the cyanide process in the metallurgy of gold and silver was more firmly established by the events of the past year, so that in considering advances in the treatment of gold and silver ores, it is practically unnecessary to go outside the progress in cyanidation. Chlorination apparently lost ground, for near the close of the year the only remaining important chlorination mill in the United States announced a suspension of operations, at least temporarily. The full significance of the announcement does not appear, but it may mean the final abandonment of chlorination as a competitor of cyanidation.

Along chemical lines the most important item in cyanidation was the announcement of the principles of the Clancy electrochemical process, and the completion and operation of the first mill to adopt the same. In brief, Mr. Clancy's proposal includes the use of cyanide, with an amide such as calcium cyanamide, and an alkaline haloid such as potassium iodide, together with electrolysis of the pulps during extraction. The net effect is the decomposition of the cyanamide into cyanide, and the regeneration of cyanates, sulphocyanides, etc., into available cyanide; while cyanogen iodide is formed and constantly renewed during electrolysis. The main object sought is a cheaper source of more active solvents for the precious metals.

The announcement of the process created a great deal of interest, and resulted in many experiments by metallurgists throughout the country. Opinions as to the feasibility of the process on a large scale were almost as various as the number of experimenters. Each found obstacles of different kind, as all the conditions for successful experimentation were not known. There was also a frank doubt expressed regarding the possible effect of complex secondary reactions at the electrodes, it being felt that the reactions would not proceed with the simplicity outlined by the inventor.

At the close of the year, however, we were furnished with some details of operation of the first plant actually using the Clancy process. It is handling Cripple Creek sulphotelluride ores without preliminary treatment in the way of roasting or concentrating. The other Cripple Creek cyanide mills employ concentration to eliminate the sulphotelluride mineral, only the tailings being treated by cyanide. It is in the direct chemical treatment of this class of ore that the Clancy process is expected to show material improvement. From results obtained to date it is expected that the operating cost will be little more than $1 per ton, including 12 cents per ton for chemicals. The tailings are showing a value of about $1 per ton. The current density employed is about 7 amperes per square foot of anode surface.

An important point emphasized last year was the necessity of considering the phase rule in cyanidation. It is well known that the solvent action of a cyanide solution diminishes as the point of equilibrium between potassium cyanide, potassium-gold cyanide, and undissolved gold is reached. This point having been reached, dissolution of the gold ceases, and the extraction cannot be materially increased even on longer agitation and the addition of fresh cyanide. The most logical conclusion is that the removal of the solution containing potassium-gold cyanide is essential, and that the substitution of a fresh cyanide solution may then materially increase extraction. Hence the advisability of designing plants so that the ore pulp may be treated with successive applications of cyanide solution, the latter being removed from the system as chemical equilibrium is approached or reached.

This leads to the highest development of the counter-current system of cyanidation, with continuous agitation and decantation, in which the richest ore is in contact with the richest cyanide solution, while the ore most nearly impoverished is treated with barren cyanide solution of high solvent power. This idea of continuous treatment is gaining ground rapidly, and, in the minds of many metallurgists, is tending to become standard practice.

Agitators of the central-tube type, in which either air or mechanical agitation is used, have been adapted to continuous systems very successfully. The vital point to be observed is the steady transference of a uniform quantity and quality of pulp from tank to tank, so that classification of sand is avoided. If this condition is not observed, difficulties will be experienced on account of coarse sand accumulating in the tanks, while slime and fine sand alone passes through the system. The only sure way of avoiding classification is to transfer a portion of the pulp overflowing the central lift-tube, using a cutting vane for this purpose.

The Parral agitating tank made its appearance last year as a rival of the now well-known Pachuca tank. Economies are claimed in the items of capacity, cost of treatment, and efficiency of air agitation and extraction. In the construction of the Parral tank there is no definite ratio to be observed between height and diameter, as in the Pachuca. It has four or more air-lifts placed at uniform distances from the center, with discharge pipes arranged so that a swirling movement is imparted to the contents of the tank.

The A-Z agitator is still popular in Australia, but has made no progress in this country.

The treatment of slime is probably the most important mechanical problem in cyanidation, and in this respect filtration shares the importance of agitation. While there is still rivalry between different forms of filters of the same type, the field of usefulness of the different types is becoming pretty well settled. For pulps composed mainly of coarse and finely ground sand, with little colloid matter, the pressure filter is superior, giving thick, permeable cakes. With pulps containing an excess of clayey or colloidal material, it is doubtful if pressures greater than those obtainable by vacuum are of advantage, and vacuum filters are generally used and recommended for this work. A recent investigator has announced that, in his opinion, the thin-cake, continuous vacuum filters are an improvement over the thick-cake, intermittent machines.

For precipitation of gold and silver from cyanide solutions, zinc dust unquestionably is growing in favor, and the later large installations have been of this type. The advantages over the use of zinc shavings are in general a lower original cost of zinc, a lower consumption of zinc, a higher grade of precipitate, a more accurate check on mill recovery, and less chance of loss by theft or fire.

An innovation only recently announced is the use of electrolytic oxygen to counteract the formation and effect of reducing agents in mill cyanide solutions, particularly those in

plants treating sulphides, tellurides, etc. In such plants the cyanide solution loses its dissolving power, even though the cyanide strength of the solution be kept up to standard. If silver predominates, the effect is still more marked. Various oxidizing salts have been used to overcome this difficulty, but have proved inefficient in many cases. A striking example of the efficient use of electrolytic oxygen is given in an article by Mr. Parmelee elsewhere in this issue. The application of this idea is expected to prove of exceptional value in the cyanidation of concentrates.

The Present Status and Prospect in the Metallurgy of Zinc and Lead.

Now that the cyanide-chlorination gold-extraction controversy seems to have been pretty definitely settled in favor of cyanide—at least for the present—the subject sure of most immediate interest and response in any gathering of metallurgical men is the treatment of complex sulphide ores. A study of the patent-office records of the world will show the same activity and interest. Several things seem fairly clear from such a study: First, no universal method has yet been discovered; second, no mechanical method can hope to become universal; third, nothing like a complete study of the situation has ever been published, if made; fourth, many metallurgical proposals and experiments are being made by men evidently unfamiliar with the handling of ores, solutions, fusions or gases in quantity; and fifth, there must be large quantities of these ores awaiting treatment.

Of these five conclusions, probably only the last needs comment. Mr. Ingalls' recent statement that most of this complex ore tonnage exists not in large deposits, but scattered in many small mines, is undoubtedly true, and greatly affects the problem. The successful process ought to be as suitable for use by the small mine as is concentration or cyanide. We once saw a man in Nevada profitably working his mine and mill single-handed. He drilled and broke his own $12 oxidized ore, shattering it as much as possible with the powder. He wheeled it out of his own tunnel, dumped it over his own screen, sending the coarse to the dump and the screenings to eight whisky barrels, where his own cyanide solution worked while he slept. This is of course a most extreme case, but it represents the ideal from the standpoint of the small mine owner.

It seems improbable that any such ideal will ever be within reach of the small producer of complex sulphides. Lead and zinc are cheap, heavy metals, which must be handled in tons, not ounces. They cannot be exchanged at the mint for coin, but must be sold on the metal market in competition with similar metals produced cheaply on a large scale. This means organization and co-operation, at least so far as sales are concerned, and if the history of mining in America teaches any one thing clearly, it is that small mine owners will not organize and co-operate.

The larger metallurgical concerns are spending large sums on this complex ore problem. They are much more likely to succeed than is the isolated inventor or small mine experimenter. Most of the reasons—such as command of capital, technical and business ability, resources in the way of plant, varied ore supply, etc.—are obvious enough, but there are several other reasons not quite so evident. One is what may be called market knowledge. Another is prestige, the weight of a good name and known achievement back of a process. Still another is knowledge and control of proper publicity, with power to get it when needed or to avoid it. A fourth is the elimination of risk. Any metallurgical process installed at a single mine assumes not only its own risk of failure (because of its own defects or weaknesses), but also assumes the mining, financial, and commercial risks of the mine itself, such as failure or change of ore supply, lack of proper capital, poor management, etc. Many a metallurgical experiment has been written down a failure which might have had some measure of success except for these contingent risks.

The larger mining and metallurgical concerns are in a much better position to estimate, minimize and discount these risks than is an individual or a small mine. This does not mean discouragement of individual effort—it means that individual effort is more likely to succeed when properly directed. If it be admitted that the owner of a small deposit of complex ore has slight chances of producing his metals in whisky barrels, or in a small plant of any sort, then his hope should be that a local market may be provided for his ore as mined. In several places, notably Wisconsin and Colorado, such a market exists and is being extended. It cannot be said to be entirely satisfactory, but the business is only some ten years old, and a safe growth must be a slow one.

Correct chemical or metallurgical theory does not insure the success of a process. A metallurgical success is not necessarily a commercial success. Improper location of plant, lack of suitable labor, adverse freight rates, and similar conditions all make for failure, but so also do such factors as lack of market for some one or more by-products, lack of selling organization for commercial products, and the necessity of meeting trade customs. This last item is rather more serious than might be supposed, and needs attention because many zinc-lead processes propose to offer some one or more chemical compounds for sale to various established industries. The amount of time and patience and skill and nerve and salesmanship and money it takes to break through an old-established trade custom, to introduce a new product or even an old one from a new source, or made in a new way, can only be appreciated by men who have tried it, and they are usually too tired and "grouchy" to care to tell about it in print.

It is of course impossible in any note of this length to touch on processes individually. All that can be said is that they are many in number, and some of them are constantly gaining ground, so that each year seems to show a better chance for the ore producer than did the preceding year. The revolutionary method which will do for lead and zinc what Bessemer converting did for copper, or cyanide did for gold, has not yet appeared. Of the possibilities of electrochemical methods we speak in a separate note.

One thing should be done for the benefit of the many workers along this line. The history of the various attempts at handling these ores lies in uncollected fragments scattered through the note-books, periodicals, technical society journals, patent office and court records of the entire world. Some of it has never been put on paper. The actual results obtained, the reasons therefor, and the details of apparatus and manipu-

lation are not to-day accessible to the metallurgical investigator, who is accordingly forced to cover much ground already proved barren. In view of the importance of the problem, the need for an authentic compilation of scattered material is evident. When it is undertaken, we hope that every interested engineer and operator will contribute liberally from his stock of knowledge.

Iron and Steel in 1911.

In a commercial review of the iron trade one must draw a clear distinction between the swings in the actual consumptive requirements of the country and the swings in demand upon the mills for material. As a rule these do not coincide, and the divergences frequently create interesting market history when without them the developments from month to month and from year to year might prove wearisome and the history of the events quite insipid.

As a rule, many of the products of the iron and steel industry are not in form for direct utilization, for ultimate consumption. When they pass outside the iron and steel industry proper, they must, usually, be fabricated before actual consumption or utilization in the daily life of man occurs. The few forms which are susceptible of final application in the form in which they leave the industry—wire nails and wire fence, for instance—generally pass through the hands of jobbers and retailers.

Thus it occurs that as regards a very large tonnage of the industry's product there are two streams, one from the producers to jobbers and manufacturing consumers, and another from these into actual consumption. The intermediary storage reservoir, the stocks in the hands of jobbers and manufacturing consumers, is subject to wide fluctuations of level, and in connection with these fluctuations much interesting market history is made, and, indeed, most disturbance is created.

The actual consumption by man in his ordinary vocations does not vary greatly from year to year, though there is an almost constant increase from year to year, interrupted or retarded only by periods of real and prolonged general industrial depression. The variations in production of such material, however, are wide, due to increases and decreases of these stocks held in intermediate hands. The mills do not carry stocks to any extent.

Two striking exceptions from the class of material just mentioned are to be noted. The railroads buy directly and in a strict sense do not carry stocks. Their consumption, regarded as the wearing out of material, varies but slightly, for the total ton-mileage in freight and passenger-mileage in passenger traffic does not fluctuate widely by any means, nor does the need for replacement of bridges, stations, etc., arise other than gradually. Nevertheless railroad purchases do fluctuate violently from time to time. Another exception to be noted is that of permanent works like office and hotel buildings, toll bridges and the like, in which definite blocks of capital are invested at a single operation, in the expectation of a steady annual return being derived.

The buying, in the case of these two exceptions, is regulated by general financial and business conditions, by the views entertained by railroad men and capitalists as to whether prices of iron and steel are likely to go up or down and whether business conditions will grow better or worse. These influences, however, are largely those which control the action of the jobbers and manufacturing consumers, who allow their stocks to decrease, or who increase them, according to their personal views.

Thus it is that in such a household article, such a durable article, as iron in its various forms we have such wide fluctuations in market conditions. A year ago, in reviewing iron and steel in 1910, we observed that in February pig iron was being produced at the rate of 31,000,000 tons per year, but at the close of the year at the rate of only 20,000,000 tons per year. No such decrease occurred in the actual ultimate consumption or utilization. In the year 1911 we have had nothing like so wide a fluctuation in production, yet we have had fluctuations which were not due to fluctuations in the rate of final consumption. The fluctuations due to changes in sentiment have been more moderate, yet they have been important, and at the close of the year we have a change, due to sentiment, which is particularly noteworthy. However crude the comparison by calendar years may be, a good starting point is furnished for discussion, and we, therefore, set our estimate of pig iron production in 1911, 23,700,000 ± 100,000 gross tons, in comparison with the officially reported outputs of previous years: 1902—17,821,307; 1903—18,009,252; 1904—16,497,033; 1905—22,992,380; 1906—25,307,191; 1907—25,781,361; 1908—15,936,018; 1909—25,795,471; 1910—27,303,567; 1911—23,700,000.

It will be observed that 1911 fell decidedly below four of the five preceding years, 1908 being the exception, with a very small output. The above is the history of production. The history of actual consumption is altogether different, both as to volume and as to the character. In 1905-6-7 there was the ordinary consumption of iron and steel in the ordinary daily life of men, somewhat heavier than formerly because business conditions were better, because population was larger and because iron and steel were being employed in new uses. In addition to this ordinary consumption there was very large consumption through railroads investing fresh capital in improvements and extensions, and through large financial undertakings being carried through, like the erection of office and hotel buildings, bridges, manufacturing plants, etc. Mill deliveries, particularly in 1907, were very slow, and jobbers and manufacturing consumers found it necessary to carry large stocks, or they would be unable to conduct their business properly.

After the panic of October, 1907, the consumption by the investment of distinct blocks of capital came to an almost complete stop, while the intermediaries who carried stocks drew upon them instead of upon mills, finding no difficulty in securing quick delivery of such special descriptions of material as they required. During this year prices were maintained by agreement, which naturally encouraged the policy of reducing stocks to a minimum. In February, 1909, prices were allowed to break, and comparatively low levels were reached. Stocks were at a minimum and as buyers started to replenish them prices were advanced. Such was the temper of the buying trade that advancing prices encouraged, rather than discouraged, the laying in of stocks, and early in 1910 stocks were as much above the normal as in the preceding spring they had been below normal. Before these stocks were reduced to

normal proportions the end of 1910 was reached. In the twelvemonth ending July 31, 1910, production of pig iron was 29,800,000 gross tons, but there is not the least ground for assuming that consumption was nearly as large. The average production of 1909-10 was 26,500,000 tons of pig iron. The year 1909 opened with very light stocks, and the year 1910 closed with normal stocks, and allowing for this fact we may take the average consumption of 1909-10 at between 25,000,000 and 26,000,000 tons, in terms of pig iron, which we find was the average production and consumption in 1906-7.

There was practically no change in total tonnage. There was, however, an important change in the character of the consumption. Railroad buying, and the investment of large blocks of capital in individual enterprises, materially decreased, while on the other hand the ordinary consumption increased.

In the comparison of pig iron production by years, 1911 suffers, but we find the year opened with normal stocks in the hands of intermediaries, but closed with practically no stocks, and allowing for this we find that actual ultimate consumption in 1911 was represented by about 25,000,000 tons of pig iron, the year not differing greatly from other years in the recent past.

The productive capacity at the present time, not the combined estimate of blast furnace managers, but an estimate based upon actual performance when the industry last ran at full capacity, with allowance for the new capacity since created, is about 34,000,000 tons, with steel making and finishing capacity in proportion. Thus we find the industry with about one-third more capacity than it has lately needed, but we find also that while total demand has not increased in five years, the ordinary demand has increased materially, so that if on top of the increased ordinary demand there should come a special flow of capital into large enterprises such as occurred in 1905-6-7, full, or nearly full, employment would be found for the existing capacity.

No such development lies in the immediate prospect. In the closing weeks of 1911 there has come a greatly increased buying movement, attended by much enthusiasm and optimistic expressions by leaders in the trade. This movement is to be studied in the light of what we said at the outset. There has been no important change in the flow of material into actual ultimate consumption; winter is the last time for such a change. Stocks in intermediate hands, however, had reached an extremely low point, and the flow from mills into such stocks is increasing. There is being repeated on a small scale what occurred in 1909 on a large scale, a buying movement after a liquidation in stocks, but there is no reason to suppose it will be carried to the disastrous length of the 1909 movement. Production of pig iron in the past three months has averaged between 24,000,000 and 25,000,000 tons a year, and this rate promises to increase by a few per cent, perhaps ten, in the early part of the new year.

The Non-Ferrous Metal Markets.

The foundations of our modern civilized society are laid on the metallurgical and chemical industries. Indeed, it has been well said that the consumption of sulphuric acid measures the degree of culture and refinement of a country. Strikingly apparent to the man in the street is the fact that we are living in an age of metals. The twentieth century is the golden age, the bronze age, and the iron age rolled into one. The progress of civilization is accordingly interdependent with metallurgical expansion. The metal markets show strength when business waxes, and weakness when business wanes. Last year the general tone of all metal markets was faltering and full of perplexity. This year there is a general air of quiet confidence throughout with, in places, an exaggerated yet in a measure real buoyancy. Each of these phases in our own business corresponds closely with the general business conditions of the United States.

The metals at present are in a strong underlying statistical position. Although statistics are notoriously unreliable, yet viewed and interpreted carefully from a practical standpoint they are a good guide. What constitutes actual consumption is largely a question of definition. When stocks pass from the refiner's hands to the warehouse of the jobber, they are held by some to enter technically into "consumption." But when they pass from the jobber to the retailer, they are surely one step further; as to the bearing of this on the iron and steel industry we speak in the preceding editorial. In the final analysis, metals do not really undergo consumption until they are placed in some useful and permanent form as a copper wire strung on a line and used as an electrical conductor to give a specific instance.

Using this the most rigid definition of consumption, we believe that in the past four years the consumption of all articles and particularly that of metals has been great, far greater than the statistics indicate. Just as in the winter months, even in a country like Ontario covered with snow the water runs through many rivulets to the streams, so in times of depression there are many hidden currents of consumption. To this intangible consumption which has been going on steadily in the past four years, the present strength of the metal markets has been due in large part. Low prices stimulate purchases especially by those who can be called economically wise. At the same time rising prices also stimulate purchasing by the doubtful ones. There has been in the past year much of each of three kinds of buying. We have in mind one central station company that has bought two years' supply of copper wire on a 12-cent copper market. We also have in mind another concern that did not buy until lately when the price for copper increased. For even with the concentration of capital and the unification of management, the laws of economics operate in much the same way that Adam Smith described 150 years ago. To our mind they always will operate, though the diffusion of intelligence both technical and financial tends to preserve a balance and to restrain both excesses of elation and excesses of fear. This is the great contribution of applied science to the twentieth century.

With regard to the particular non-ferrous metal markets but little need be said. All have shown great signs of strength, especially the copper market. For as we said last year, in this age of electricity, there cannot long be too much copper. The new producers of copper have not reached the producing stage as fast as anticipated nor have they produced the quantity expected. These facts coupled with large exports and a good domestic trade have resulted in a strong position for copper.

Spelter has been most uncannily strong. At present its market can be said to be a nominal one. The discoveries at Fryer Hill, Leadville, have resulted in the shipment of large tonnages of carbonate. Improvements in ore dressing such as the Huff electrostatic separator have further increased the tonnage of low-grade zinc ore. This, with the decrease in the number of active retorts due to the exhaustion of the "Iola" gas pool, has made the condition of the smelter quite favorable. For, with a restricted smelting capacity and an increased tonnage of low-grade ore, the smelting companies can do about as they please both in purchases of ore and in sales of metal.

Lead has averaged but little higher in 1911 than in 1910. Its price is so largely controlled by one producer that it fluctuates slowly. The shortage of lead ore is still a problem. Nickel and "monel" are both controlled by one concern, but their production has largely increased. Tin is the only metal not produced to any extent in this country. The price is largely manipulated, but it is only fair to state that it is not produced fast enough for the needs of the world. There is no telling what will happen in the tin market. Silver, of course, is a coin metal and does not fluctuate as do the base metals. At the close of the year its price is slightly higher than a year ago.

It cannot be expected that this increase in the strength can continue for long. In the first place, high prices tend to restrict actual consumption. Next there is a large excess of producing capacity in the metallurgical works of this country, and a continuation of the rise in prices will cause the rehabilitation of abandoned plants and also stimulate the erection of new and improved plants. A pseudo-prosperity or a boom founded on the supposition that our country is ready for a period of inflation would be the worst thing imaginable for the metallurgical industry. But a sane expansion on lines proportionate to the growing needs of modern life is probable. On the other hand, the diminution of doubt and the absence of perplexity is at least gratifying. When the workers are in the foreground and the shouters are in the background, there is little cause for worrying. Such, indeed, is the pleasant condition of the metal business at the opening of 1912.

The Patent Reform Problem Before the American Institute of Chemical Engineers.

The rules which restrict carefully and materially the membership of the American Institute of Chemical Engineers are usually explained on the ground that they are the foundation of the prestige of the Institute. This is so evidently true that another advantage is usually overlooked. It is that this very restriction of membership greatly facilitates intense work on important questions of public policy. The patent reform problem is a case in point. It was the subject of a most interesting discussion at the meeting of the Institute held last week in Washington.

Three formal and very able papers dealt with the subject, the first by United States Commissioner of Patents Moore and the other two by two eminent lawyers, Mr. Edmonds and Mr. Kenyon. A lively discussion followed, in which, among others, Dr. Hill, of the United States Patent Office; Mr. Rogers, president of the Washington Patent Association, and Dr. Baekeland, the new president of the Institute, participated. Besides this symposium of papers the members of the Institute had an opportunity to study during one afternoon the details of the working of the Patent Office, and it may not be amiss to refer here first in a few words to the impressions made on the visitors on this occasion.

The work of the United States Patent Office is simply amazing, even to those who are acquainted with our patent system but have no occasion to come in contact with the tremendous and painstaking work involved in all details. One of the striking things is the card index of chemical literature brought together by Dr. Hill and his many assistants, a work of such magnitude that it has taken them many years of diligent work and that it is still consuming their full attention. This card index numbers several millions of cards; it is open free of charge to anybody interested in chemical literature. Anybody engaged in chemical research work, whether the latter be related to patent work or to scientific research, can find there in a much shorter time than anywhere else whatever has been accomplished on any chemical subject. Many a chemist engaged in a search of chemical literature prefers to take the trouble of a trip to Washington to find there quickly all references instead of spending weeks and months running from one library to another. It would be highly desirable if the Public Library of New York had a duplicate of this card index, but the work would involve a rather large outlay of money, which could only be obtained by special donation for that purpose.

The immense increase of the work of the Patent Office due to the ever increasing number of patents has made the old building absolutely obsolete—a regular fire trap where invaluable records may be lost at any time should a fire occur, and the latter risk is imminent on account of the hopelessly overcrowded conditions. It is a pathetic sight to behold the inadequacy of this building and the excessively crowded places in which the examiners and assistant examiners have to perform their work. Yet the United States Patent Office is the only paying department of the country, and has now an accumulated balance to its credit of over seven million dollars. Why not tear down that old and antiquated Greek mausoleum and build a spacious, well ventilated, well lighted, modern fireproof building with part of the seven million dollars and do justice to the inventive genius of the nation and the industries which depend on it?

However, what is more needed than a new building is reform in the patent system—not so much in the patent laws themselves as in the rules of practice of these laws. We venture to express the conviction that the United States patent law is good enough, perhaps better than that of any other nation. But when it comes to settle suits for infringement the whole proceedings become so unnecessarily cumbersome, long drawn, expensive and undecisive that the word *absurd* is really much too mild for it.

This condition of affairs gives an immense advantage to the very rich, whether they are individuals or corporations, as they can draw out a suit indefinitely provided they are willing to stand the expenses attached thereto; it puts the inventor or manufacturer of small means at a hopeless disadvantage. The most needed reforms seem to be the following:

Firstly, a patent court of appeals which can decide definitely all appealed cases, while now the circuit courts often disagree, so that in several cases a patent has been declared valid in one circuit and invalid in another, this leading to hopeless confusion and unsettled decisions.

Secondly, in case of infringement testimony should be taken in open court and before a judge so as to do away with the absurd practice now prevailing that irrelevant testimony can be used indefinitely and without the slightest restraint so as to pad the records and prolong indefinitely the taking of testimony and thus postpone the issue.

Thirdly, the attempt should be made to reduce the unnecessarily large number of appeals in cases of interference, and thus do away with the advantage that a wealthy contestant may have in an interference trial before the patent is allowed making it more burdensome than an infringement suit.

There are many other reforms proposed, but the above mentioned seem to us the principal ones, and unless action is concentrated on the most important ones it is very likely that attention will be scattered again to such a point as to produce no practical result, and that the matter will be further postponed to the detriment of the industries of the nation.

It behooves all scientific and technical organizations to take an active part in this patent reform because conditions have become well-nigh intolerable. The public at large is more interested in this reform than it knows. For after all the public has to pay indirectly but surely the bill of the expensive lawsuits and the uncertainties of patent decisions.

Unfortunately the engineering societies of this country have remained rather torpid and indifferent. There will be no trouble in getting the needed patent reform if those who represent the industries of the country show that they want it The American Institute of Chemical Engineers, the Washington Patent Law Association, the Inventors' Guild, and the American Bar Association are up till now the only organizations which have taken some live interest in the matter. The American Institute of Electrical Engineers is just beginning to do so. Why do the American Chemical Society, the American Electrochemical Society, and the many other engineering societies not take more interest in this vital question?

Electrochemistry and Chemistry in 1911.

While yearly reviews of progress in electrochemistry, written some ten years ago, used to be almost of a sensational character—a new electric-furnace reaction often meant the foundation of a whole new industry or the revolution of an old one—it is different now. Electrochemical processes are no longer in the test-tube stage. The electric-furnace plants of Niagara Falls, the electrolytic copper refineries of New York harbor, the electrolytic caustic and chlorine plants, the aluminium industry—all are now well established old industries working quietly and showing from year to year regular sane and safe commercial advances in output and conquests of the market. Of course, commercial success is not everything in life, but even pure idealists must concede that success is a test of inherent value. From this viewpoint the present quiet and eminently satisfactory position of the established electrochemical industries is clear evidence of the soundness of their foundation.

This fact speaks well for the prospects of electrochemical methods in metallurgy which now attract so much attention. There are many of them of greater or smaller scope in their respective fields. In steel metallurgy the electric furnace is making steady progress. In this country the progress has been somewhat slow—certainly too slow for those who expected a cataclysm of the old order of things—yet the pace has been just what was to be expected from a sane evolution along strictly logical lines. The crucible-steel makers continue to install electric furnaces for high-class steels. At the close of the year the first electric furnace has started operation in this country in foundry practice—especially for automobile castings. But with respect to the biggest problem—the combination of Bessemer converter and electric furnace for the manufacture of steel for rails, etc.—commercial finality has not yet been reached, though steady technical progress has been made, especially in the baking of the very large electrodes required for the large furnaces.

In gold metallurgy there are electrolytic amalgamation, still in the process of evolution, and the Clancy process combining the use of calcium cyanamide and electrolysis as novel features and about which the report reaches us while going to press that the first commercial installation at the Ajax Mine in Victor, Colo., is a decided success. How manifold the methods of electrochemistry are, however, is best seen in connection with the complex zinc-lead sulphide problem. For its solution three essentially different processes are being proposed and are awaiting the commercial test in the near future. Firstly there are the different electric-furnace processes, of which at least three can no longer be said to be in "the infancy of the experimental stage." Secondly, dry chlorination along lines formerly followed by Swinburne and Ashcroft, to be tested in two new plants in Colorado (Malm process) and in Montana (Baker-Burwell process). Thirdly, at San Francisco a plant is going up to test the Reed-Hering process embodying electrolysis of sulphate solution with recovery of sulphuric acid. A more detailed discussion of this whole very interesting and complex situation must be reserved for a later issue.

What was said above about the quiet and eminently satisfactory working of the electrochemical industries may be said of the chemical industries in general in this country. However, there was one glorious main event of the past year, a milestone in the history of chemistry in America—the opening of the new Chemists' Club building in New York. It was wonderful to conceive the idea of a Chemists' Building as a monument representative of the united democracy of American chemists. It was more wonderful to realize this idea without outside help through the combined efforts of American chemists. But the most wonderful miracle is the outcome of the crucial commercial test of the first year—the fact that the Chemists' Building Company made both ends meet and did even better. If anybody has to refute again the still too prevalent idea that idealists are necessarily poor business men, the Chemists' building is classical evidence.

This augurs well for the success of the big event of the coming year—the first International Congress of Applied Chemistry to be held in America. It must be made, and will be made, a success worthy of American chemistry.

Washington Meeting of the American Institute of Chemical Engineers.

The fourth annual meeting of the American Institute of Chemical Engineers was held from Dec. 20 to 23 in Washington, D. C., in accordance with the program given in full in our last issue.

Dr. Leo H. Baekeland, of Yonkers, N. Y., was unanimously elected president for this year, succeeding Dr. Frerichs, of St. Louis.

Several very interesting papers were presented. The most important feature of the professional sessions was a symposium on Friday, Dec. 22, on the needs of the reform of our patent system. United States Commissioner of Patents E. B. Moore opened the subject by an interesting paper, followed by two able and suggestive addresses by Mr. Walter D. Edmonds and Mr. R. N. Kenyon. A very active discussion followed in which Dr. Hill, of the United States Patent Office; Mr. Rogers, president of the Washington Patent Association; Dr. Baekeland and many other members took part.

L. H. BAEKELAND, PRESIDENT AMERICAN INSTITUTE OF CHEMICAL ENGINEERS.

The afternoon of the same day was devoted to a visit to President Taft. Then followed a visit to the Patent Office. Of other excursions that to the Indian Head Proving Grounds proved particularly interesting, occupying the greater part of Thursday. As permission to visit the Proving Grounds is difficult to obtain, and societies have thus far been refused admission, it was an exceptional privilege extended to the Institute.

The banquet at the New Willard was a very entertaining event and well attended. Dr. Chas. F. McKenna was a brilliant toastmaster, introducing successively retiring president Dr. Frerichs and President-elect Dr. Baekeland. Further speeches were made by United States Patent Commissioner Moore, Rear Admiral Twining, Colonel Dunn, Dr. Stratton, director of the Bureau of Standards, and Dr. H. W. Wiley. Dr. Hillebrand, of the Bureau of Standards, was also one of the invited guests.

On the whole the meeting was a really fine success. A full report of the papers presented at the meeting is reserved for our next issue.

Program of London Meeting of the Institute of Metals

The London meeting of the Institute of Metals will be held at the Institution of Mechanical Engineers, Storey's Gate, Westminster, S. W., on Tuesday, Jan. 16 and Wednesday, Jan. 17, 1912.

The meeting will commence on Tuesday afternoon when the president-designate (Prof. W. Gowland, F. R. S.) will deliver his inaugural address on the subject of "Copper and Its Alloys in Early Times." In the evening of the same day the annual dinner of the Institute will be held.

The whole of Wednesday, Jan. 17, will, if necessary, be devoted to the reading and discussion of a number of interesting papers, among which may be mentioned the following: "Properties of certain copper alloys at high temperatures," by G. D. Bengough; "Further experiments on the inversion at 470° C. in copper-zinc allovs," by Prof. H. C. H. Carpenter; "The influence of oxygen on copper containing arsenic or antimony," by R. H. Greaves; "The nomenclature of alloys," by Dr. W. Rosenhain; "Poisoned brass and its behavior when heated in vacuo," by Prof. T. Turner, and a paper by Dr. Carl Benedicks, of the University of Stockholm, dealing with some novel experiments on a zinc-antimony alloy. The latter paper, the second to be contributed by a foreign member of the Institute in the past three months, will be unique in the literature of British learned societies, as it will contain illustrations in color reproduced by autochrome (natural color) photography. This is a novelty also in this country.

Mr. G. Shaw Scott is the secretary of the Institute of Metals. The offices of the secretary are at the Caxton House, Westminster, S. W., London.

Chemists' Club.

At the yearly election, held at the Chemists' Club in New York City on Dec. 12, the following officers were elected:

Dr. Morris Loeb, president; Mr. Otto P. Amend and Dr. George D. Rosengarten, vice-presidents; Dr. John E. Teeple, secretary; Mr. William James Evans, treasurer.

MORRIS LOEB, PRESIDENT CHEMISTS' CLUB.

The three club laboratories, named in honor of Wolcott Gibbs, Robert Bunsen and August W. Hofmann, are now as completely equipped with distributing mains as those of any college; the vacuum and compressed-air plants have recently been installed. These laboratories are equipped with all the necessary fixtures for work. Apparatus is not supplied, but arrangements for the latter can be made on very short notice without leaving the building. These laboratories can be engaged for periods varying from one week to three weeks.

The Unveiling of the Wolcott Gibbs Bust at the Chemists' Club.

Of the many memorable events which have already taken place in the new stately building of the Chemists' Club in New York City, none has been more pleasingly touching than the unveiling of the bust of the late Professor Wolcott Gibbs on the evening of Nov. 25.

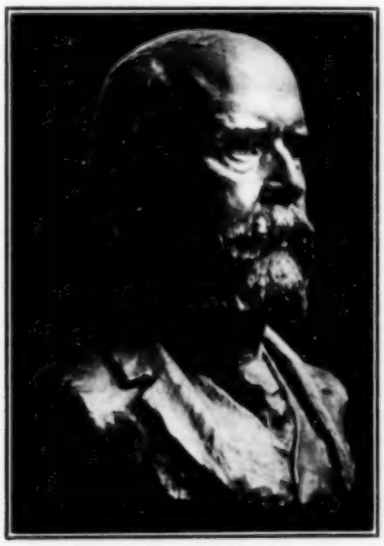
The club members and their guests, including a large number of ladies, assembled in Rumford Hall at 8:30 under the presidency of Dr. Russell W. Moore.

Mr. Alfred W. Gibbs, on behalf of the donors—nephews and nieces of Professor Gibbs—made a short address in presenting the life-size bronze bust, which is the work of the well-known sculptor, Mr. Ernest Wise Keyser.

President Moore in a few aptly chosen words accepted the bust on behalf of the Chemists' Club, whereupon Dr. Charles E. Munroe in an eloquent address (printed in full in *Science*

of Dec. 22, 1911) reviewed in detail the life of Gibbs, his characteristics and his work as a teacher. He emphasized the fact that Gibbs was the founder of the U. S. Sanitary Commission, the first scientific organization of its kind, and out of which the Red Cross movement developed. Gibbs was also one of the founders of the Union League Club and of the Century Association, the former the great patriotic and the latter the important scientific organization of New York.

Wolcott Gibbs' scientific work was analyzed in extenso and the importance of his work on the complex acids and complex bases was clearly brought before the audience. It was remarked that the work of Gibbs and Geuth, on the ammonia-cobalt bases, published by the Smithsonian Institution, was a classic in this complex field of inorganic chemistry.



BUST OF WOLCOTT GIBBS AT THE CHEMISTS' CLUB IN NEW YORK.

Dr. Morris Loeb, in moving a vote of thanks to the family for their generous donation of the bust, and to Dr. Monroe for his brilliant address, read a letter from Prof. Charles Loring Jackson, the oldest living colleague of Professor Gibbs in the chemical department of Harvard University.

Prof. Charles Baskerville, of the College of the City of New York, in seconding the vote of thanks, called attention to the fact that New York can with pride point to the time when Gibbs was professor of chemistry at the Free Academy, now the College of the City of New York, where he was followed by Ogden Doremus.

These exercises were followed by a musical entertainment by a triple quartet from the Columbia University Glee Club—a fitting climax to the dignified and edifying memorial, since Columbia claims Dr. Gibbs as an alumnus.

Then followed some most excellent instrumental music by the "Chemists' Club's Own" trio from the Music School Settlement.

The Chemists' Club may well be proud of this evening, which will long be remembered by all who had the privilege of attending the simple and beautiful exercises.

The Western Metallurgical Field.

Cyaniding Base Ores and Concentrates.

Elsewhere in this issue are given some details on cyaniding in Clear Creek county, Colorado. Reference is made to certain former experiments made in this general district, including Gilpin county, and of the technical success of the same. In this connection the expression of Mr. M. D. Draper, who made the experiments referred to, will be of interest. Mr. Draper's remarks appeared in the Colorado School of Mines *Magazine*.

"Together with several associates, I made a large number of cyanide experiments on the ores of Gilpin county about two years ago, the various tests and experiments covering a period of about one year. It was found that there was no particular difficulty in the cyanidation of the ores; and crude concentrates containing 2 per cent copper and only 10 to 12 per cent silica could be cyanided direct without undue cyanide consumption and with a high extraction of values. In fact the average extraction in a series of sixteen parallel tests, after the proper conditions were ascertained, was found to be 87 per cent. Another point of considerable interest was that the average extraction of several Gilpin county custom milling plants was found to be good, considerably better than has generally been supposed and, in brief, probably in excess of 70 per cent. Notwithstanding this fact, the shipper seldom receives a net profit of more than 50 per cent of the gross value of his ore, and the difference has been found to be due to the freight and smelting charges on the concentrates.

"There is no doubt whatever in my mind, that the establishment of a central milling plant in Gilpin county, with a treatment charge of $2 per ton, would result in unparalleled activity in this pioneer gold district. This plant would use the ordinary present methods of amalgamation followed by concentration, but would differ in cyaniding not only the tailings but the concentrates as well. The present situation, speaking of Gilpin county as a whole, is that the shipper is unable to work at a profit if the grade of the ore is under $6 per ton. In many cases this statement is equally true as to a head value of $7 per ton. A treatment charge of $2 per ton would insure a 300-ton cyanide plant a handsome profit, and the shipper would receive some net return from an ore running as low as $3 per ton, though this figure would not allow for mining cost.

Amalgamated Copper Company and Porphyry Properties.

The decision of the Amalgamated Copper Company to engage in the operation of what are commonly termed the porphyry copper mines, has a wide significance when viewed from several angles. It is Amalgamated's first venture in a field which has hitherto been considered unattractive to them. But the apparent success of other similar ventures, and the possibility of materially adding to the quantity of copper refined and sold by the Amalgamated interests, has probably put the matter in a new light, and has made a consolidation of properties near Miami, Ariz., seem very attractive.

The purchase of the Live Oak by Mr. John D. Ryan consummated a deal which the Miami Copper Company had endeavored to effect, but without success. With the Live Oak in his control and the Inspiration in friendly hands, it appears that negotiations are now pending for the purchase of the Miami, or at least for the consolidation of all three properties into a gigantic copper company under Amalgamated direction. The Miami already has a large mill in operation, and a plant for the Inspiration has been under consideration. It is estimated that with all three properties producing, Amalgamated could add 75,000,000 pounds of copper annually to its already premier production in the copper world.

Another point of view is that the merger would be of benefit to the Raritan Copper refinery, which is owned by the International Smelting & Refining Company, an Amalgamated concern. The addition of 75,000,000 pounds of copper annually to be refined at this plant is a considerable item, and would be a welcome source of income for the International company which has been spending immense sums for construction during the past two or three years.

It is pointed out that conditions for a merger of the Miami, Inspiration and Live Oak properties are most favorable. They are contiguous and have a large tonnage of ore developed which will average about 2 per cent copper. The exploration work on the Live Oak and Inspiration is supposed to have demonstrated a life of twenty-five years at an annual production of 50,000,000 to 75,000,000 pounds of copper per year. When the Inspiration mill was originally talked of, it was calculated that it would produce about 50,000,000 lb. annually. No statement has yet been issued indicating the probable terms of a consolidation, but a merger appears imminent.

The Search for Potash.

The fact that the imports of potash salts into the United States in the calendar year 1911 will have a value of about $15,000,000, is stimulating both government and private enter-

prise in searching our country for possible sources of potash. The matter is of such importance that the government has appropriated $20,000 to enable the U. S. Geological Survey to make a systematic search, cooperating with the Bureau of Soils which has an appropriation of $12,500. At present this money is being expended principally in drilling in favorable locations, notably in the beds of the prehistoric Lakes Bonneville and Lahontan in Nevada. One well known as the Timber Lake Well is to be pushed to a depth of 1000 feet in an effort to locate beds of potash salts. This well is in the bed of Lake Lahontan, as that lake is known never to have overflowed.

Another source of potash which is being investigated is the brines and bitterns from salt works. Many samples of these waste liquors have been collected and are being analyzed, with the result that some of them are showing considerable percentages of potash salts. One natural highly concentrated bittern has been investigated which shows the presence of potassium chloride associated with lime and magnesia, the separation of which offers a nice problem in chemical technology.

More recently, however, workable deposits of alunite have been discovered, and private interests are investigating their treatment. Alunite is a hydrated sulphate of aluminium and potassium containing 38.6 per cent SO_3, 37 per cent Al_2O_3, 11.4 per cent K_2O and 13 per cent H_2O. On heating, this mineral gives up its water and the sulphuric acid, which is combined with alumina, leaving a residue which is 55.6 per cent of the original mass, and of which one-third is soluble potassium sulphate and two-thirds insoluble aluminum oxide. This mineral offers also a possible source of aluminium.

A large deposit of alunite has been discovered near Marysvale, Utah, a portion of which is said to have passed into the hands of the Armours, who are taking out a 100-ton test lot. The mineral is pink in color and has been locally known as pink spar. The deposit has been located in former years as a possible source of metallic value in gold and silver, but no analyses were made and its true value was not known until recently. This is but another example of the advisability of analyzing minerals which are found in abundance in certain places.

Copper Smelter Construction.

On December 2, 1911, the first train load of copper ore from the Nevada Douglas mines was delivered to the new Mason Valley smelter, Thompson, Nev. Although the smelter is not in commission, the receipt of ore points to early operation, and to the increase of copper production in Nevada. The copper mines in the Mason district are the only ones of prominence among those which have been developed in the past few years, in which the ore occurs in vein formation as distinguished from the disseminated deposits of the porphyry mines. The smelter is reported to be one of the best planned and constructed plants in the West.

The Hayden, Ariz., smelter of the American Smelting & Refining Company, which is to treat the concentrates of the Ray Consolidated Copper Company, is being pushed to completion as rapidly as possible. The original site of the plant, on which the Ray company had begun construction, has been abandoned, and new work commenced at another point. This will delay completion somewhat, but it is hoped to have the plant ready for the receipt of Ray concentrates by the stipulated time, April 1, 1912. The power plant and furnaces are being designed for the use of oil as fuel. Two reverberatory furnaces are being built for the immediate treatment of the Ray ore, and no plans are as yet made for the handling of custom ore.

Company Reports.

The third annual report of Stratton's Independence, Ltd., gives some interesting data on the work of the company's mill at Victor, Colo., where the old mine dump and some low-grade mine ore is treated by concentration and bromocyanide. In the fiscal year, July, 1910, to June, 1911, the tonnage treated was brought up to the capacity figure of 10,000 tons per month, having averaged slightly more than that amount in March, April, May and June, 1911. The ore milled during the fiscal year under consideration averaged in gold content, 0.157 oz. gold per ton, which, for the gross tonnage of 109,800 tons milled, gives a total of 17,288.96 oz. gold.

There was recovered in concentrate	7,546.80 oz.	or 43.65%
There was recovered in bullion	4,814.80 oz.	or 27.85%
Total recovery	12,361.60 oz.	or 71.50%

The costs of operation in the last three months of the fiscal year, in which the mill reached its estimated capacity of 10,000 tons per month, were as follows:

Milling cost	$1.2390 per ton
Mining cost (dump)	.0891 per ton
Total	$1.3281 per ton

These figures for recovery and cost are remarkably in accord with the estimates of Mr. Philip Argall, manager, made in 1907 on the completion of his tests of the dump ore. At that time he estimated that, on a monthly tonnage of 10,000 tons, an extraction of 70 per cent might be obtained at a total cost of $1.52 per ton treated. Of this total cost he estimated $1.24 per ton for milling and $0.10 for mining. The report states that it is expected to reduce the total costs by 7 cents per ton during the current fiscal year. The working profits from mine and mill have slightly exceeded the estimate of $10,000 per month, and although the mine has not shown any permanent improvement during the year, it is thought that with the mill running at full capacity a profit of $10,000 per month can be continued.

The Goldfield Consolidated Mines Co., Nevada, treated the largest tonnage in the history of the company in the month of October, 1911. The ore production amounted to 30,626 tons, of an average value of $28.09 per ton. This was milled with an average extraction of $26.77 per ton, or 95.31 per cent. The total net profit realized was $616,141.87, or $20.12 per ton. The distribution of the cost, $6.80 per ton, was, as follows:

Mining		
Development	$0.83	
Stoping	2.04	$2.87
Transportation		.08
Milling		1.96
Marketing		.74
General expenses		.38
Bullion tax		.31
Construction		.46
Total cost of operation		$6.80
Miscellaneous earnings		.15
Net cost per ton		$6.65

The construction of the concentrate residues treatment plant is progressing satisfactorily.

Non-Ferrous Metal Market.

The closing month of 1911 witnessed activities in the metal market which gave a more optimistic outlook for the future of some of the non-ferrous metals. Copper and spelter reached a high point, and there was more active speculation, which, however, was based on a feeling that higher quotations are likely to prevail for some time. The quotations on tin fluctuated somewhat, but there was a decided shortage of spot tin, and premiums were demanded. Lead shared the general rise in price.

Copper.—The most important advance which has occurred for a year took place in this market during December. Most of the purchases were for export, but a large volume of domestic business was reported. The last quotations available are 13½ to 13¾ cents for Lake, and 13.45 to 13.50 cents for electrolytic.

Tin.—Early in the month the price was as high as 45 cents, with little business transacted, but later the price dropped to 43.75 cents for December tin, New York. Sales were mostly for future delivery, and little business was done in spot metal.

Lead.—Since our last report the price of lead has been advanced, and the latest quotations have remained unchanged for some days at 4.45 cents, New York, and 4.30 to 4.35 cents, St. Louis.

Spelter.—December deliveries of this metal were made at advanced prices, and as high as 6.75 cents was reported paid, although the highest quotations were nominal. Later, prices fell off, and the last quotations are still nominal, with little business transacted. The market has apparently settled around the following quotations: New York, 6.15 to 6.27½ cents; St. Louis, 6 to 6.12½ cents.

Other Metals.—The aluminium market has been a little more active, with prices slightly higher in New York, viz., 18¾ to 19 cents. Sales of antimony have been larger and the market is firmer. Prices range from 6⅝ to 7¾ cents for various brands. Prices for quicksilver are lower; $44 per flask of 75 lb. at New York, and $43.50 at San Francisco.

The Iron and Steel Market.

December proved to be even a better month in the iron and steel trade than was foreshadowed in our review of a month ago, when it was observed that November opened with gloom universal and closed with sentiment uniformly optimistic. December saw a definite and marked improvement in buying and a slight advance in prices, while it was made certain that production would be increased at the beginning of the new year.

Following the fairly heavy contracting for finished steel products in November there was very heavy contracting in December, and at the same time orders and specifications for early delivery became much heavier. The effect of increased specifying was cumulative, as had been expected in case the start should be made, for all buyers, whether jobbers or manufacturing consumers, had allowed their stocks to drop almost to nothing, and were depending for the ordinary conduct of their business upon the quick shipments which the mills could make upon new specifications. As soon as the mills fell a trifle behind it became necessary for buyers to place additional specifications, to order in advance of requirements, and as such business piled up in mill books, the period of delivery fell farther behind, inducing additional specifying. Thus, from being able to make shipment of many products in from two days to a week in the early part of November, the mills close the year with actual specifications involving from two to six weeks' steady run, while in some cases it is believed that with only a moderate amount of fresh tonnage from week to week full operations can be maintained for from four to six months.

A curious feature of the reaction in prices was that the lowest prices in the entire movement were as a rule found late in November, so that the slight advances which have occurred are the more impressive by comparison. Late in November shading in merchant pipe became fairly general, and the situation was recognized by the National Tube Company issuing a new list under date of December 1, reducing butt weld pipe by one point, or about $2 a ton, and lap weld by two points, or about $4 a ton. At the same time wire prices declined, to a basis of $1.50 for nails.

The advance started with an announcement on November 29 by the Republic Iron & Steel Company that its prices on bars were advanced to 1.10 cents for prompt delivery and 1.15 cents on contracts for first quarter and half, these prices being $1 above the existing market. Other bar manufacturers at once concurred. On December 11 the American Steel & Wire Company announced an advance of $1 a ton in wire products, making nails $1.55, and the independents immediately followed. Next day the American Sheet & Tin Plate Company informally advanced its prices on sheets to minima of 1.90 cents for 28-gage black, 2.90 cents for 28-gage galvanized and 1.40 cents for 10-gage blue annealed. On 28-gage these were advances of $3 a ton from the lowest price done on prompt delivery, and $2 a ton from the lowest done on contracts for the first quarter or half of the new year. The sheet market stiffened at once, although it required some time before prices of 1.85 cents on black and 2.85 cents on galvanized disappeared entirely. At about this time plates and shapes definitely advanced to a minimum of 1.15 cents, the market having previously stiffened by concessions from the open quotation of 1.10 cents disappearing. Bars later advanced to this level, and a few mills advanced their regular asking price on plates and shapes to 1.20 cents.

Making comparison with the lowest prices that were done on particularly attractive orders, the average advance on the whole list of finished steel products outside of rails has been about $1.50 per ton, while the advance in openly quoted prices is fully $1 per ton.

Normally December is a dull month, and phenomenon of activity was probably due to the extremely low prices ruling, with the encouragement furnished by the large car orders recently placed, indicating that railroads, usually slow to buy in a low market, were ready to take advantage of bargain prices.

The net declines in steel products from January 1, 1911, to the low point at the close of November were as follows: Bars, 1.40 cents to 1.05 cents, $7 per ton; plates and shapes, 1.40 cents to 1.10 cents, $6 per ton; merchant pipe, 79 per cent off to 81 per cent off (on ¾ to 1½ in.), $4 per ton; wire products, from $1.70 to $1.50 for nails (advances in the early months to $1.80), $4 per ton; sheets, from 2.20 cents to 1.75 cents occasionally done and 1.80 cents freely done, $8 to $9 per ton; tin plates, from $3.60 to $3.30 on large and $3.40 on small lots, $4 to $6 per ton (with a $3.70 market ruling from February to August) and billets from $23 to $18.50, $4.50 per ton.

Pig iron declined an average of about a dollar a ton during the year.

Pig Iron.

The Youngstown Sheet & Tube Company bought 25,000 tons of Bessemer iron for January and February delivery from the Ohio Iron & Steel Company, at the regular price of $14.25, furnace, other small sales of Bessemer being made at the same figure. The Jones & Laughlin Steel Company bought about 30,000 tons of basic iron for early delivery to its Aliquippa plant, intending to start the steel department, never before operated, without starting the blast furnaces. There are four 250-ton Talbot furnaces. This iron is understood to have gone at special prices, averaging about $12.10. A few sales were made in the month at $12.25 and $12.30, quotations advancing late in the month. In late November and early December a moderately heavy buying movement occurred in Southern iron, chiefly at $9.50 and $9.75, Birmingham, the market later firming up to $10 asked for extended delivery, but occasional lots for prompt shipment being available at $9.75. In New England territory there was fairly heavy buying, at the expense of prices. Prices stand quotable as follows: Delivered Philadelphia, $14.75 to $15 for No. 2 X; f. o. b. Birmingham, $9.75 for prompt and $10 for forward delivery, No. 2 foundry; f. o. b. valley furnaces (90 cents higher delivered Pittsburgh), Bessemer, $14.25; basic, $12.50; No. 2 foundry, $13; forge, $12.50; malleable, $12.75 to $13.

Steel.

Early in December there was heavy covering of both billets and sheet bars, consumers who had previously bought for a month at a time taking December and January deliveries, and in some cases later deliveries also. For the earliest deliveries billets went as low as $18.10 to $18.50 and sheet bars at $19, f. o. b. maker's mill. As buying progressed prices advanced,

and the market closed the month at $19.50 to $20 for billets and $20.50 to $21 for sheet bars, f. o. b. Pittsburgh.

Finished Material.

The year closes with the following prices firmly maintained, f. o. b. Pittsburgh unless otherwise stated:

Rails, standard sections, 1.25 cents for Bessemer, 1.34 cents for open-hearth, f. o. b. mill except Colorado.

Plates, tank quality, 1.15 cents to 1.20 cents.

Shapes, 1.15 cents to 1.20 cents. Steel bars, base, 1.15 cents.

Iron bars, 1.25 cents, Pittsburgh; 1.25 cents, Philadelphia; 1.15 cents to 1.20 cents, Chicago.

Wire nails, base, $1.55; plain wire, 1.35 cents, base.

Black sheets, 28-gage, 1.90 cents; galvanized, 2.90 cents; blue annealed, 10-gage, 1.40 cents.

Perkin Medal Award to Herman Frasch.

At the meeting of the Perkin Medal Committee (consisting of representatives of the Society of Chemical Industry, the American Chemical Society and the American Electrochemical Society), held on December 15 at the Chemists' Club in New York, the Perkin medal for 1912 was unanimously awarded to Mr. Herman Frasch for his chemical engineering work in the sulphur and oil refining industries.

The formal presentation of the Perkin medal to Mr. Frasch will take place on January 19 at the regular meeting of the New York section of the Society of Chemical Industry.

CORRESPONDENCE.

Carbon Monoxide and Dioxide.

To the Editor of Metallurgical and Chemical Engineering.

Sir:—I note with interest a communication from Mr. Carl Hering in the December issue of your journal regarding the nomenclature in use at the present time to some extent for defining CO and CO_2.

I want to reiterate what Mr. Hering says in this respect and would recommend that the two gases be specifically designated by the names Carbon Monoxide and Carbon Dioxide. This would leave absolutely no room for doubt or misunderstanding.

I have run across the names lately in several cases when the CO has been termed carbonic oxide, and the CO_2 called carbonic acid much to my disgust and discomfort. This has been in cases of gas analyses which I had occasion to compare with analyses made at the present date, particularly in reference to wood gas, where the content of CO is always rather in evidence. The use of the names carbonic acid and carbonic oxide has caused me a great deal of useless hesitation and figuring, and I suppose my case is not alone in this respect.

I hope that this matter will have the attention of the chemists of the country, to be arranged to better advantage.

Memphis, Tenn. James C. Lawrence.

The Position of the Induction Furnace.

To the Editor of Metallurgical and Chemical Engineering:

Sir:—Referring to Mr. T. D. Robertson's article in your November issue on "The Grönwall Steel Refining Furnace," I wish to comment on his remarks on the induction furnace. He says that "it would appear that the induction furnaces are on a decline." Further on he makes the statement that Messrs. Grönwall, Linblad, and Stalhane "having originally worked with various types of induction furnaces without great success, decided to design an arc furnace."

Even if Messrs. Grönwall, Linblad, and Stalhane have not aided the induction furnace for steel melting and refining, it is hardly proven that induction furnaces are on the decline, except perhaps in Mr. Robertson's opinion.

That the genius of practical, electrical steel men does not always run in the same direction is evident from Messrs. Grönwall, Linblad, and Stalhane's self-confessed failure to produce a successful induction furnace for steel melting and refining; whereas Rodenhauser produced a successful melting and refining induction furnace after many years experience with Kjellin induction furnaces. The Röchling-Rodenhauser electric induction furnace was only commercially built about four years ago, yet to-day there are seventeen built and three building, all of commercial size. This is only one type.

The Kjellin type has produced twelve or more commercial size furnaces and as many more of smaller sizes. There are still other types of induction furnaces.

The writer saw nine different R. & R. and other electric furnaces this summer. One of the R. & R. furnaces is of 10 tons capacity and at the time was turning out tool steel in 10-ton lots, by refining basic bessemer metal with phosphorus 0.08 and sulphur 0.09 and getting the P and S down to 0.008 and sometimes to traces, at the same time achieving a high degree of deoxidation. This refining was accomplished with from 200 to 250 kw-hours to the ton, with a total cost of $4 to $5 for 2204 lb. over the cost of the Bessemer metal. These R. & R. furnaces have been known to melt cold stock with 580 kw-hours to the ton, showing a combined electrical and thermal efficiency of 85 per cent without a residue of hot metal in the furnace.

Surely this is not a decline. The decline is tilted the other way.

New York City. C. H. Vom Baur

Electric Energy from Cohesive Forces.

To the Editor of Metallurgical and Chemical Engineering:

Sir: In some recent electrolytic tests in which the metal was dissolved from solid anodes and deposited on the cathode in a loose, finely divided form, the writer noticed that the voltage required was considerably greater than what it was thought it ought to be, because when as much metal is dissolved as is deposited the voltage required for chemical purposes is zero, leaving only that required to overcome the resistances, which should be small.

Its rather large value in this test therefore suggested that it might be due to the fact that the metal was being finely subdivided during the process of electrolysis; this comminution must consume some energy, as the forces of cohesion of the molecules must have been overcome by some larger force, and apparently the only way in which the equivalent of that force can manifest itself in the circuit, if it originates there, is by a counter emf, as it does in the case of an electric motor. Hence the energy of these cohesive forces might in that case be measured electrically.

If this argument is correct, it might follow conversely that if the anode were made of finely divided metal, say a precipitated metal, and the cathode deposit were in a dense form, like ordinary metal, a direct emf would be formed. Hence such an electrolytic cell when short-circuited might possibly generate its own current, if the process is reversible. If so, it would be a case of producing electric energy by the cohesive forces of matter.

The writer did not have the time or facilities to make further investigations. The above should, therefore, be considered merely as a suggestion to those who may have the facilities, time and interest to make the researches.

It recalls another case suggested by the writer many years ago, namely, that when metal is carried from the anode to the cathode between two horizontal plates it should require a less voltage when the anode is the upper one than when it is the lower one, and the difference should correspond to twice the energy required to raise the weight of metal deposited through the distance between the plates. It might also be possible that such a cell, short-circuited, would produce a current. An alternating current passed through such a cell should show a slight direct-current component.

Philadelphia, Pa. Carl Hering.

The Japan Excursion of the American Institute of Mining Engineers

By Jos. W. RICHARDS, Vice-President of the Institute.

The American Institute of Mining Engineers has added one more to the list of its famous and most successful excursions. Of more than local significance, this last pilgrimage has taken on a distinctly international complexion, and the spontaneous welcome accorded by the Japanese has given it truly national importance.

Voyage to Japan.

In the November issue of this journal (page 564) an account was given of the San Francisco meeting which lasted from Oct. 10 to 16. As we stated then, the meeting was in reality a fête, a festival of good things not to be forgotten.

On Oct. 17 eighty-three members and guests of the Institute left San Francisco on the steamship *Manchuria* of the Pacific Mail S. S. Company, bound for Japan, there to be guests of the forty Japanese members of the Institute.

Honolulu, in the Hawaiian Islands, was reached Oct. 23, and a most interesting day spent visiting the sights in and around that city. Most of the party took a 25-mile automobile ride, dined at the Maona Hotel on the famous Waikiki beach, and some even tried the surf riding in catamaran canoes. Under the present government régime the islands appear to be making vast strides in material and educational progress, and everyone seems to be well satisfied with the result.

The stretch of 3500 miles from Honolulu to Yokohama was covered in ten days of beautiful semi-tropical sailing. The voyage was enlivened by frequent entertainments.

Dr. R. Kanda, of Japan, who had come over to America for the express purpose of accompanying the visitors across the Pacific, gave three interesting lectures on Japan, the country, the people, and the language. Mrs. C. H. Sleeper lectured on "Life in Manila."

Mr. D. W. Brunton described "The Laramie Tunnel," a paper eliciting a spirited discussion and comparison of European and American tunneling practice. Mr. W. L. Clark opened up an afternoon's discussion of the proposed "New Mining Code," in which everything from Alaska coal to labor unions was discussed. The writer of this report occupied an afternoon describing the new electrochemical industries of Norway and Sweden.

Among the lighter amusements were a fancy dress masquerade ball and a complimentary dinner extended to the Engineers by the steamship company.

On Nov. 3, several hours ahead of time, the steamer reached Yokohama, and the real goal of the trip, the making acquaintance with Japan and the Japanese, began then *instanter*.

PARTY AT NIKKO COPPER REFINERY.

There are forty members of the Institute in Japan and they were one and all enthusiastic in their reception of the eighty-seven visitors; add to this nucleus the hearty co-operation of over 200 others who also belong to "Who's Who" in the mining world of Japan, and the no less cordial interest of many men high in affairs of state, and the nature of the welcome accorded can be more easily imagined than described.

Baron Shibusawa was chairman of the reception committee, which acted in a very graceful and generous manner, Dr. Wada (an honorary member of the Institute) was prominent upon it, and with T. Dan of the Mitsui Company, R. Hotta of the Mining Institute of Japan, R. Kanda, Prof. B. Katsura of Kyoto, R. Kondo of the Oshio R. R., K. Kato of the Tokyo Chamber of Commerce, K. Nambu of the Associated Mine Owners, Engineer K. Noro, M. Otagawa of the Furukawa Mining Company, Prof. W. Watanabe of Tokyo, and S. Yamanouchi of Takata and Company, constituted the general reception committee.

SMELTING WORKS AT SHISAKA ISLAND.

Dr. R. Kanda was sent by this committee to San Francisco to greet there the visitors and to accompany them on their journey—a courtesy which was highly appreciated, for Dr. Kanda was in position to give the party most valuable and

useful information concerning Japan on their seventeen-day journey across the Pacific.

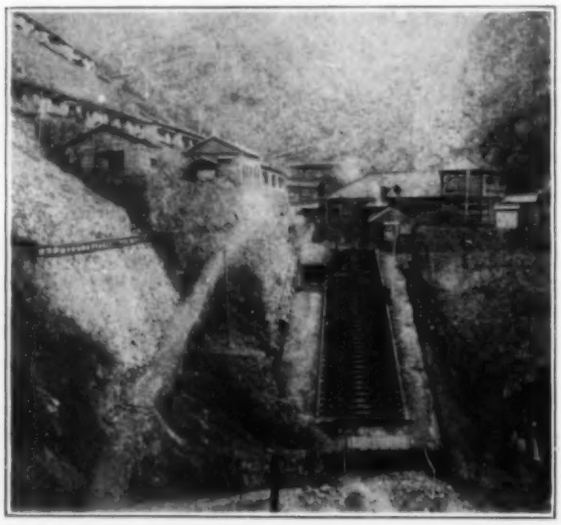
After a memorable day spent in Honolulu and ten more days of comfortable steaming toward the setting sun, the "Manchuria" anchored in Yokohama harbor early on the morning of Nov. 3—the Emperor's birthday. The first rays of the rising sun gilded the snowy summit of sacred Fuji, clearly visible in all her glory, 40 miles away, and when its later rays showed the vessels in the harbor gay with bunting and the city in festive array the scene was charming and the impression ineradicable.

BARON KICHIZAYEMON SUMITOMO.

Representatives of the reception committee were on the steamer early and the round of pleasure had begun. To describe in detail all of the numerous receptions accorded the party would be impracticable, yet so spontaneous were they and so cordial that each one deserved special mention.

In general, the procedure was as follows: The visitors were met at the landing place or railway station by the local committee, the mayor and other officials of the town, also a host of the populace, and an address of welcome was made. The party was then conducted in automobiles, carriages or jinrickishas, gaily decorated, to the town hall or a company's office, where amid gay decorations refreshments were served, badges of welcome distributed and more formal speeches of welcome given and responded to.

The gentlemen then divided into parties to visit near-by mines or industrial works, while the ladies were further entertained by visits to temples, gardens, and the fascinating shops.



CEMENTATION PLANT NEAR THE THIRD ADIT MOUTH, SUMITOMO COPPER MINE.

Comfortable, even luxurious, hotels were found everywhere we stopped, and the evenings were usually devoted to a banquet, followed by dances by "Geisha Girls," and frequently a display of fireworks. At times every one was supplied with flags and decorated with flowers in the enthusiasm of the final moments.

Such were the general features of the reception everywhere accorded, and often the smaller the place the larger the welcome. Ten different badges of welcome were distributed and the mint at Osaka struck off artistic memorial medals in silver as permanent souvenirs of the trip.

The general outline of the eighteen days' travel in Japan was as follows: First reception at Yokohama; two days at Tokyo, with official reception; a day at Kyoto, the ancient capital; visit to Nara; three days on steamer on the Inland Sea, covering visit to Shisaka copper smelting works, the sacred island of Myajima, the government steel works at Yawata and the towns of Moji and Shimonoseki; one day at the port of Kobe; a day at Osaka; two days again at Tokyo; two days at Nikko, including visit to the Nikko electrolytic copper refinery of the Furukawa Company; a day at Kamakura; a day in the mountains at Myanoshita, near to grand old Fuji; and a final winding-up day at Yokohama before sailing for home on the "Siberia."

Side trips were taken by smaller parties from Moji to the great Miiki coal mines and from Nikko to the Ashio copper mines and smelter. The noteworthy particulars of the various visits, reporting them from the technical point of view, are as follows:

Shisaka Island Smelter.

This is a copper smelter belonging to the Sumitomo firm, and located on an island in the Inland Sea. The house of Sumitomo, with headquarters in Osaka, has been in the mining and metal business for nearly two centuries; the present head of the firm is Baron K. Sumitomo, a young man of forty-seven years who looks barely thirty. The firm runs a bank with fifteen branches; its industrial enterprises comprise a coal mine in Kyushu employing 2500 men and furnishing 800 tons of coal daily and coke for the smelting works, the Besshi copper mine, the Shisaka smelter, a metal-working plant in Osaka employing 900 hands manufacturing sheets, rods, bars, tubes and wire of copper, brass, bronze, zinc and aluminium, as well as insulated wire and electric cables, and a steel foundry with 600 hands, using one basic and one acid-lined open-hearth furnace, making wheels, axles, gears, cylinders, ore crushers, dredging machinery, ship machinery, etc. The policy of this firm of thus manufacturing into final form its own crude material is to be highly commended as the wisest industrial policy for the enrichment and development of Japan.

SHEET MILL, SUMITOMO COPPER WORKS.

Coming back to the mine and smelter the visiting engineers visited only the latter, but a few words may be said about the mine. The Besshi copper mine was discovered in 1690 and has been continuously worked since then by the Sumitomo firm. It is in the mountains, 3000 to 4000 ft. above sea-level and 12½ miles distant from the seacoast. The ore deposit is a mile long and 2 to 25 ft. wide, consisting of copper pyrites and pyrite contained in crystalline schists.

The daily output averages 600 tons of 4 per cent copper ore, carrying 35 per cent sulphur, 35 iron, 15 silica, 7 alumina, 1.5 lime, and 1 magnesia.

The mine water, estimated to carry 180 tons of copper a year, is run over scrap iron to precipitate "cement" copper, and the waste is run into the sea by a 10-mile long ditch in order to protect the intervening agricultural lands. The precipitate obtained by means of this process carries 80 per cent copper.

A central water-power plant is being constructed, 25 cu. ft. of water per second being available, with 2000 ft. fall, which will produce 3000 kw.

The smelter was originally on the coast near to the mine, but the smelter gases damaged neighboring farm lands to such an extent that the government appointed a commission to study the question and make recommendations. This commission, on which were Prof. W. Watanabe, of the Mining Institute of Japan; J. Yokobori, professor of metallurgy in Kyoto University; Y. Watanabe, professor of metallurgy Kyushu University, and M. Oshima, professor of metallurgy in Tokyo University, besides experts in agriculture and forestry, recommended the removal of the smelter.

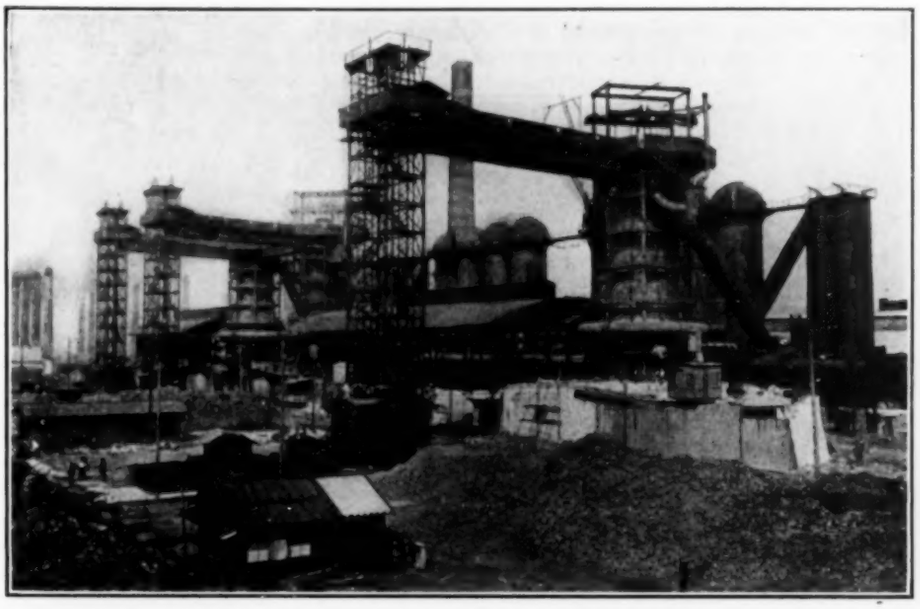
As a consequence in 1901 the works was moved to Shisaka island, 9½ miles from the coast in the Inland Sea. Even at this distance the smelter gases (containing about 2 per cent of SO^3 and 1 per cent of SO^3 at the stack) damage the young vegetation, and it is customary to reduce the running of the smelter two-thirds for twenty days in April or May to save the young rice, and a like period in August or September to protect young barley.

The smoke and fume question is a universal bugbear and the solution adopted by the Sumitomos is indeed instructive.

As for the island smelter itself its location was visible 10 miles across the sea from the cloud of white smoke. The island is entirely occupied by the works and the workmen's village. The population turned out *en masse* to receive us, their interest being partly accounted for by the statement that no foreign white ladies had ever before visited the island; many of the inhabitants are born, live and die there, without once leaving it. Fireworks and frequent *banzais* testified the heartiness of the welcome.

The average laborer is paid 30 cents per day; women, boys and girls receive as little as half that. It should be mentioned, however, that the company supplies houses nearly rent-free, and furnishes supplies at their wholesale cost, sometimes even cheaper for such staples as rice and sugar.

A fleet of 180 lighters brings ore, coke and supplies from the mainland. Numerous stalls of the Freiberg pattern are used for roasting. Five smelting furnaces are working, four with 10 to 13 per cent of coke on ordinary smelting and one with 3 to 6 per cent of coke operating by a semi-pyritical process.



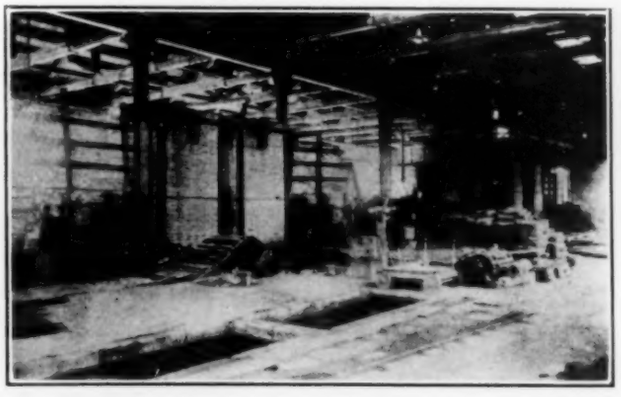
BLAST FURNACE OF THE IMPERIAL STEEL WORKS, YAWATA.

Fine ore is sintered in six pot-roasters 10½ ft. in diameter by 3 ft. deep, in which a 10-ton charge of ore, sand and flue dust is roasted from 20 per cent sulphur down to 5 or 6 per cent in 4½ hours. This finely designed plant was just starting in operation; it was designed by their able metallurgist, K. Nakamuro, with the assistance of H. Kudo, mechanical engineer.

Noteworthy were the overhead crane for lifting the pots and placing them on dumping trunnions at one end of the plant, and the use of an agglomerator to prepare the moistened charge by balling it into small pellets as it passes through a rotating conical barrel. The latter device produced pellets about as large as peas and it was stated that 30 per cent of the time of a blow was saved by putting the material into this shape. The six pots produced 360 tons of sinter per day.

The smelting furnaces are 4 by 24 ft. at the tuyeres, with settlers 10 ft. in diameter, and produce 18 to 23 per cent copper matte, average 20 per cent. The slag is granulated by a heavy stream of water and run into the sea; it averages 0.2

GENERAL VIEW NIKKO COPPER WORKS.



FURNACE ROOM, NIKKO WORKS.

per cent copper. A bank of by-product coke ovens, producing 48-hour coke, is on the island. The surplus oven gas is burned under steam boilers by an ingenious form of burner.

The matte is roasted in stalls and smelted directly to black copper. This is refined in two reverberatories and cast into ingots branded "K. S." This ingot copper carries about 0.015 per cent silver and 0.0003 per cent gold ($2.25 and $2 worth per ton respectively).

The annual output is now 6500 tons of copper, 99.7 to 99.9 per cent pure; it is estimated that the firm has produced altogether 200,000 metric tons of copper from this Besshi ore.

ELECTROLYTIC REFINERY, NIKKO COPPER WORKS.

The Sumitomo firm, like several other similar firms in Japan, gives us the somewhat novel impression of being run and managed more for the best interests of the country and people than for the aggrandizement of the firm; it co-operates with the government in minimizing damage to agricultural lands by smoke, it re-forests the mining lands, it takes a paternal interest in its employees, and it is actively fostering manufacturing industries which after all are going to be the backbone of Japanese progress toward wealth and power. We heartily wish the firm continued success in developing its broad and far-sighted policies.

The Yawata Imperial Steel Works.

This fine works, established ten years ago, situated on a branch of the Shimonoskei Strait, near to the Japan Sea, is the largest and most interesting in Japan.

The pig-iron part comprises three blast furnaces erected by Lührman, in German style, and run under the direction of

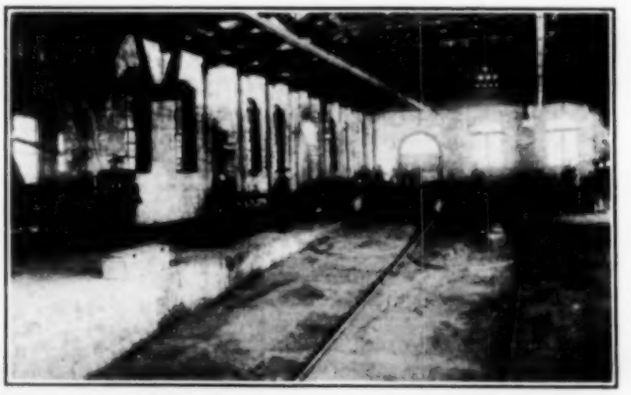
DRAWING FINE COPPER WIRE, NIKKO WORKS.

Engineer T. Kawai. Each furnace has four stoves, of which three are in constant use. The output of each is 160 to 180 tons daily.

The ore comes from Hokaido and Corea, and carries 0.5 per cent sulphur. The coke carries 1 per cent sulphur.

The chief point noted about these furnaces was the low pressure of blast, 4 to 5 lb. per square inch. We cannot but think that an American manager would find a way of driving these furnaces at a considerably greater speed.

The pig iron made is rather high in silicon and manganese, and the management fears that higher blast pressure would



ROLLING COPPER RODS, NIKKO WORKS.

still further increase these, but faster running might be so accomplished as to give less silicon; the high silicon is really the result of slow running.

A fourth furnace is contemplated, the dimensions of which will be as follows: Internal capacity, 525 cm; height, 21 m;

PRODUCTS OF NIKKO COPPER WORKS.

diameter of crucible, 4 m; diameter of bosh, 6.8 m.; estimated output, 250 tons daily; raising the total works capacity to 750 tons. The furnaces are well built, with good stove capacity, and should really be driven at considerably higher capacity.

GENERAL VIEW OF SMELTER, ASHIO.

Coke is produced at the works in one bank of Coppee ovens and one bank of Solvay ovens. The product is 600 tons of

coke daily from 1000 tons of Kyushu coal; surplus oven gas is used in the 160-ton pig-iron mixer.

The steel department contains two acid-lined Bessemer converters, which are run in the same leisurely manner as the blast furnaces, a 10-ton heat being blown in twenty-five to thirty minutes. The steel produced cast excellently and rolled

SETTLING AND FILTERING PONDS, ASHIO.

finely. There are also eleven open-hearth furnaces of 25-ton capacity each; eight more 50-ton furnaces are being built.

The foundry contains three cupolas and a 15-ton acid open-hearth furnace for steel castings; also crucible steel capacity for 5 tons per day for making special steels and high-speed tool steels. An electric furnace for replacing the crucibles is being figured upon.

With common labor paid 20 to 35 cents per day many things are possible at this plant which would be impossible in the United States, but yet steel making is strenuous work in which muscular exertion is at a large discount wherever machinery plus intelligence can be used.

The Japanese are intelligent, not only the managers but the workmen, and since the latter can be taught to handle machinery we are of the opinion that much further mechanical development at this works is not only possible but probable. Hand filling of blast furnaces should be abandoned, also dump filling of coke ovens, and many labor-saving devices now used in America could be introduced with profit in various parts of the works.

The works produces a great variety of materials, each on a rather small scale, but it has been of inestimable value in the recent industrial development of Japan. We predict for it still further improvement and still greater usefulness to the country in the future.

The coal and steel barons of Kyushu, the Pittsburgh district of Japan, entertained the visiting engineers right royally at the town of Moji, the extreme western point of the trip; everyone felt the warmth of the welcome and also felt at home in the smoky atmosphere and bustling activity of this busy neighborhood.

The Nikko Copper Works.

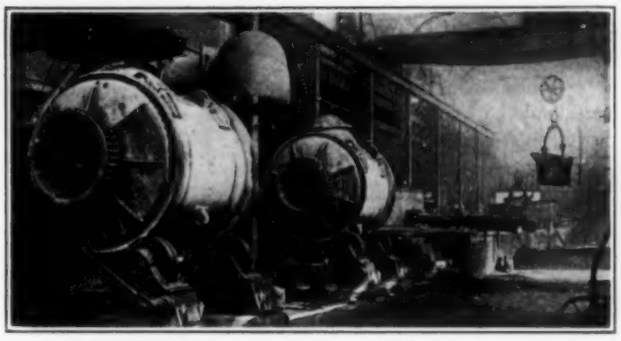
Ninety miles north of Tokyo is the most-visited spot in Japan—the famous shrine of Nikko. Here amid lofty cryptomerias, surrounded by towering mountains, the great Iyeseyu built the group of lacquer and gold temples and his impressive mausoleum which attract visitors and pilgrims from far and near.

But in addition to these standard attractions we were here the guests of the Furukawa Mining Company, which has its great copper mine at Ashio, 18 miles away, and its refining plant on the outskirts of Nikko itself.

This large company, founded only thirty-five years ago, operates fifteen copper mines, two silver mines, one gold mine, one lead mine, four coal mines, a copper refinery and a coke plant. Its working force is 22,000 men and it produces yearly 14,000 tons of copper, 700,000 ounces of gold and silver, 500,000 tons of coal and 30,000 tons of coke. The present head of the company, T. Furukawa, is only forty-seven years old; he accompanied the visitors throughout a large part of their trip.

A small company of about a dozen left the main party at Nikko and rode horseback over the mountains to see the Ashio mines and smelter. These mines were discovered in 1609, but when bought by this company in 1877 were producing only 52 tons of copper yearly; under the enterprising management of the Furukawas the production has reached 7000 tons annually. The deposit consists of a mountain about 2000 ft. high, worked from four different directions, and in which over fifty veins of copper ore are being mined.

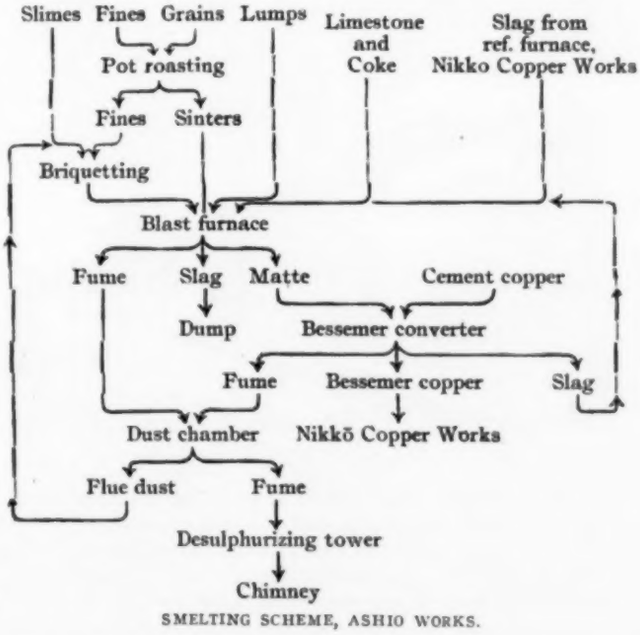
There are three large concentration plants. The mine water is run over scrap iron, producing 36 tons of cement copper monthly; the waste water, together with the slime-carrying



INTERNAL VIEW OF ASHIO SMELTER.

water from the concentrating plants is treated by a special process before discharging into the river, in order to render it harmless to the rice fields which it irrigates.

The ores are chiefly chalcopyrite, pyrite, quartz and some sphalerite and galena. They go to the smelter as lump, grain,



SMELTING SCHEME, ASHIO WORKS.

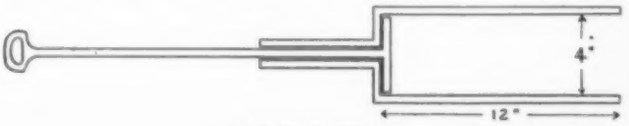
sands, slimes, cement copper and copper precipitate. About 60,000 tons of material, averaging 12 per cent copper, is smelted per year.

The precipitation of the waste waters deserves mention. Lime is used as the precipitant but when used all at once the precipitate was too poor in copper and too high in silica and alumina to pay smelting charges; by first running in a small part of the milk of lime the bulk of the iron, alumina and silica is precipitated first, while afterward the further addition of lime gives a precipitate rich enough in copper and low enough

in alumina to stand smelting. The details of this ingenious process have been worked out by Mr. Imai, who promises their early publication in the Bulletin of the Institute.

The smelter plant consists of pot roasters, rectangular smelting furnaces and Bessemer converters. The blister copper ingots are sent over the mountain by rope tramway to Nikko for refining. The outline of the plant is given in the diagram of the smelting scheme on bottom of page 19.

The following items caught our attention as worth mention and comment: The briquetting press was of novel design, patentered by S. Watanabe, mechanical engineer of the company.



CHARGING DEVICE.

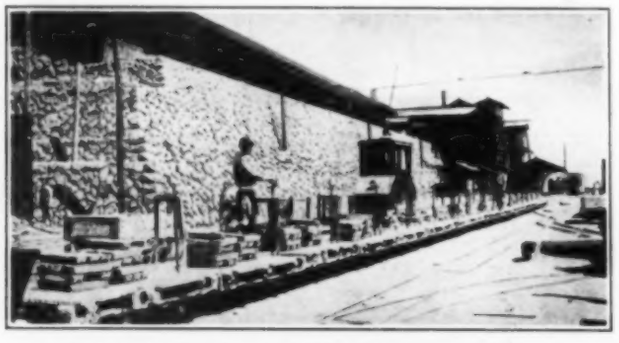
It operates like a stamp mill, each briquet being moulded by six stamp blows of 120 lb., falling 10 to 12 in. Two horsepower suffices for a daily production of 36 tons.

The briquets were dried by being piled around hot cones of slag. The machine was ingenious, very cheap and reasonably effective.

Eleven pot roasters in operation serve to prove the success with which the Japanese have adapted this method to the needs of copper metallurgy. A smelter charge taken from the black-board alongside the furnace was:

Lump ore	300 lb.
Grain ore	100 lb.
Briquets	200 lb.
Pot roast sinter	400 lb.
Converter slag	150 lb.
Limestone	325 lb.
Coke	100 lb.
Coal through tuyeres	30 lb.

The last item deserves special mention. Each tuyere has an outer-flap valve about 5 in. in diameter, in which is a smaller ¾-in. swinging valve. Through these two workmen are con-



TRANSPORTATION OF BESSEMER COPPER, ASHIO.

tinuously employed at a furnace smelting 100 tons per day in introducing soft bituminous coal into the furnace.

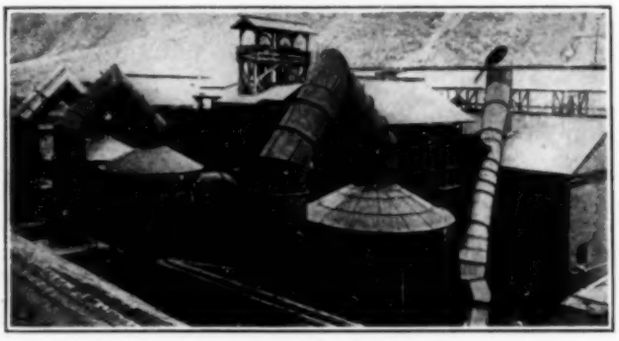
The coal is in lumps of ½ to 1½ in., and is put first into a charging cylinder 4 in. in diameter by 12 in. long, provided with a piston worked by a rod through the handle.

This implement was turned up, filled with coal, thrust rapidly into the tuyere, and the contents then discharged by pushing the handle operating the piston. (See above diagram.)

After three or four cylinders had been thus put in at one tuyere it became necessary to use a poking rod, through the small valve, to clear the tuyere and get all the coal into the furnace.

About 3 tons of coal is thus put into each furnace per 24 hours. The men's wage is 30 cents each, per 12 hour shift. The superintendent explained that this practise kept the smelting zone hotter, prevented the tuyeres from building out "cold noses" into the furnace, nearly eliminates the necessity of barring down charges from above, and increases the smelting capacity of the furnace.

The practise is common at other Japanese copper smelters; in fact it is said that at the Kosaka smelter, working ore with

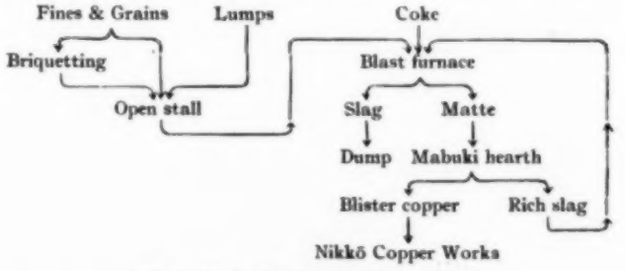


DUST CATCHER OF ASHIO SMELTER.

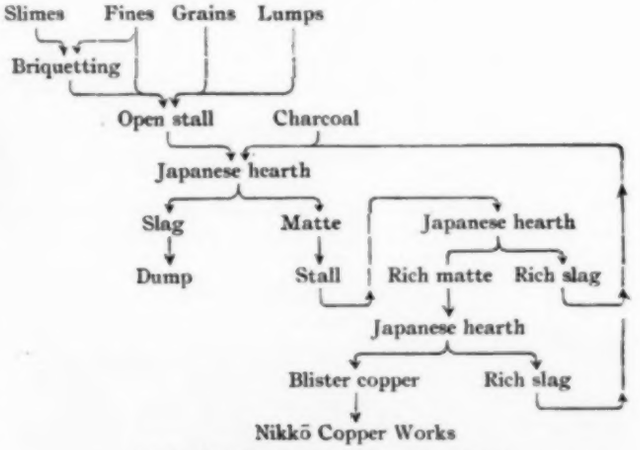
40 per cent of barite, it was found impossible to run the furnaces without this introduction of coal at the tuyeres.

It is certain that the effect is advantageous and American engineers should be able to devise an automatic stoker to put in the coal mechanically if they ever decide to adopt the practise.

The Ashio smelter now bessemerizes its matte in two acid-lined converters, which blow 10 tons of matte (in two successive charges of 5 tons each) in 10 hours, to 4 tons of cop-



SMELTING SCHEME, FUROHURA WORKS.



SMELTING SCHEME, NAGAMATSU WORKS.

per. The only noteworthy feature of this operation is that one converter lining lasts only two blows—an unnecessarily short life from an American point of view. It appears probable to us that properly directed effort could at least double the life of these linings.

Of great metallurgical interest is the fact that these converters have replaced an old Japanese process which was practically a "hearth-Bessemer" operation; it was called the "Mabuki" process or "Japanese hearth," and is still operated at

some other copper smelters, such as Furokura and Nagamatsu, diagrams of whose smelting schemes are given on page 20.

The "Mabuki" process or "Yamashitabuki" was invented at the village of Yamashita, near Tada, province of Settsu, nearly 400 years ago. The Japanese smelting hearth is made in the ground with stone walls cemented with clay and lined with brasque.

Common dimensions are 24 inches diameter by 10 inches in depth, with one tuyere 1 in. in diameter, others are 36 in. by 16 in., with 2½-in. tuyere; the charge is ½ to 3 tons of matte, using 50 to 30 per cent of its weight of charcoal.

At Furokura there are six hearths, each 32 x 15 in. and capable of working a charge to 6000 lb. of 45 per cent copper matte down to blister copper, 98.5 per cent pure, in 24 hours.

At Kusakura a similar hearth is operated with air blast and oil spray as fuel, instead of charcoal.

At Nagamatsu the hearth is used also as a smelting furnace, 5000 lb. of roasted ore being smelted to 35 per cent matte in 10 hours, using 1500 pounds of charcoal.

The slag being pulled off, water is sprinkled on the surface of the matte, which is then stripped off in thin sheets. This matte is partly roasted in stalls, and again smelted by a similar operation to 60 per cent matte.

This second matte is roasted in stalls and smelted to blister copper in the hearth.

This process, as a whole, is the Welsh process, operated in hearths instead of reverberating furnace, but where the hearth receives molten matte directly from the shaft furnace the operation is really "bessemerizing" of matte upon a hearth. The operation is an interesting metallurgical "survival" from the olden times.

The Ashio smelter furnishes one more interesting feature to the metallurgical tourist, viz., an attempt to suppress the chimney gases. The photograph of the plant shows four towers in series on the side of the mountain between the works and the chimney.

These are brick towers 12 ft. square by 42 ft. high, down which (through 200 holes of 0.4 in. diameter) milk of lime is showered. The gases average 0.8 per cent SO^2 and 0.3 per cent SO^3. By using 40 cu. ft. of water per minute and 300 lb. of burnt lime per hour, in the towers, all the SO^3 is intercepted and 40 per cent of the SO^2, reducing the latter to 0.5 per cent.

The water, containing much SO^2 in solution, is run into settling ponds, more lime there added, and the clear waste water filtered through sand before being discharged into the river. The government evidently considers this as a fairly satisfactory solution of the smoke and fume question at this particular smelter.

Between the smelter and refining works at Nikko the company has at Hosoo a 6000-kw. power plant.

The water supply of 5 cu. meters per second as a minimum comes from the Daiya River, fed by Lake Chuzenyi. The upper power house, containing one 2000-kw. unit, has 643 ft. head, the lower house, with two 2000-kw. units, has 714 ft. head. The generators are built by the Siemens-Schuckert Co., and are 11,000-volt, 3 x 109-amperes, 50-period, 2000-kw., 375-r.p.m. machines with a power factor of 0.77. The power is transmitted 13 miles to the Ashio mines and smelter and 2 miles to the Nikko refining plant. Another 2000-kw. unit is to be added.

The Furukawa Company entertained the whole party of the Institute visitors at luncheon at the Nikko works, where a remarkable reproduction of the famous red-lacquer sacred bridge of Nikko was build up of copper wire-bars, with the stream represented by blue vitriol crystals. The copper used was one day's output of the works. At Ashio the smaller party was received with great enthusiasm; an archway erected at the office gate (see photograph) had pillars of Bessemer bars, representing likewise one day's output of the works.

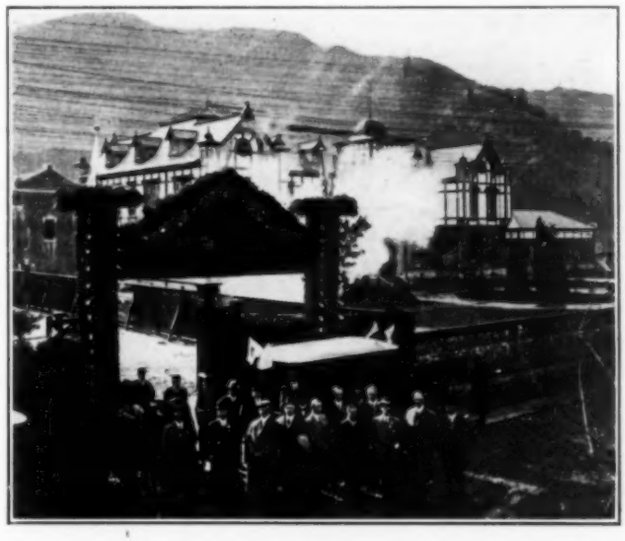
The Nikko works receives copper from the various smelters of the Furukawa Company, the blister copper from "Mabuki" hearths and Bessemer converters, and produces annually 2000 tons of ingot copper, 5000 tons of copper wire, 500,000 ounces of silver bullion, besides copper sulphate crystals.

The blister copper containing too little gold and silver to pay for electrolytic refining is melted, furnace refined and cast into commercial ingots averaging 99.7 per cent pure.

There are four refining and casting furnaces, two of which have casting machines, the capacity of each furnace being 40,000 to 55,000 lb. per heat. A run in such a furnace comprises 4 hours charging, 8 hours melting, 4 hours refining and poling, 4 hours casting. Workmen around the furnace receive 25 to 50 cents gold per day of 12 hours.

The casting machine is of Japanese design, being the invention of Mr. Kuku, mechanical engineer of the company. It resembles the simple semi-circular pig lead casting device used in lead works, except that the iron pipe is replaced by an open launder pivoted on the spot where the stream of copper falls from the furnace.

The outer end of this launder carries a separate tilting section whose outer lip can be raised while the stream is being transferred from one mould to another, the section being hung



PARTY AT ASHIO. PILLARS OF GATEWAY ONE DAY'S OUTPUT BESSEMER BARS.

upon a lever operated by the man who swings the launder. One man regulates the flow of copper from the furnace, a second moves the launder and regulates the casting, a third dumps the moulds and rights them again. The apparatus is cheap and effective and its operation very simple.

Blister copper for electrolytic refining is first furnace-refined up to about 99.44 per cent copper, and then cast into slabs 12 x 20 in., 1 in. thick. These are heated to redness, rolled down to ¼-in. thickness and then cut into pieces 12 x 24 in., weighing 20 lb. The edges are frequently curved and ragged. Two of these sheets form one plate in the refining tanks, the side edges being painted half-inch wide with asphaltum varnish.

There are 300 refining tanks, run on the series system, a current of 600 amperes being sent through four tanks in parallel. Twenty-five such sets in series are run on a 250-volt circuit, there are three such circuits. The drop is 10 volts per tank, and, with 26 electrodes per tank, this is 0.38 volt per electrode.

The current density is 38 amperes per square foot (418 amperes per sq. meter). The distance of the electrodes apart 1 to 1.5 in. (2.5 to 4 c. m.). The ampere hour efficiency is about 35 per cent. The circulating solution is 5 gal. per min. per tank. Ten per cent. of total electrolyte are purified every two weeks. The excess of dissolved copper is sold as blue vitriol.

The very low current efficiency is doubless due to the large path for current to pass below and above the electrodes and

at the center between the plates. About 10 c. m. space was allowed for accumulation of slimes below the electrodes, and as much as 3 c.m. of solution was seen in many tanks above the electrodes; the plates had curved or ragged edges, so that where they met in the center of the electrode considerable free space, probably averaging one inch, existed.

The total area of free path for current was thus some 6 in., against 24 in. of copper surface. This would easily account for 50 per cent loss of ampere hour efficiency in such a 26 electrode series tank. Part of the low ampere hour efficiency is probably also to be ascribed to running the plates until within 5 per cent of the original electrode is dissolved, which would entail redissolving and reprecipitating considerable pure copper.

This practise gives very low anode scrap to be remelted. The anode residue is removed from the cathodes by hand chipping with chisels. Altogether, the plant is interesting, and particularly so because of the room for technical improvement.

The visiting engineers certainly owe a large debt of appreciation to the Furukawa Company for the hospitality lavished upon them, for the very instructive information gained at its mines and works and for the personal attentions most numerous and acceptable received throughout the trip from the president and professional staff of the company.

Afterthoughts.

We travel to enlarge our mental horizons, to make friends, and, if possible, to help those we visit.

Not one of the engineer pilgrims to Japan but will acknowledge the gaining of much information and mental improvement from the trip. We found the Japanese the greatest eclectics of the age; they have selected from abroad and adopted with the greatest discrimination those ideas and processes which are best adapted to their conditions and needs.

In the line of mining and metallurgy Japanese graduates of Freiberg and Clausthal are numerous, and German methods were more in evidence than American. This is most intelligent discrimination, for their labor conditions and resources to work with are much more similar to those in Germany than to those in the United States. We believe, however, that as time passes and conditions change future development will more and more approximate to the methods we use.

Even in philosophy and religion the educated Japanese shows the same eclecticism and discrimination. He is fusing together the best principles of Buddhism, Confucianism, Shintoism and Christianity into a code and rule of life which, as far as high thinking and right living are concerned, may be regarded as a religion of high practical value as well as pure ethical ideals.

Japanese engineers have also frequently improved upon their teachers, so that evidences of inventiveness and originality in mining and particularly in metallurgy are frequently visible and give the professional visitor ideas to take away with him.

Friends, appreciative and sincere, we feel that we have numerously added to ourselves, to beautify our sentiments towards Japan and to enrich our lives.

Turning our thoughts Japanwards, have we any useful ideas to give them? Is our generous appreciation of their magnificent half-century's accomplishment and advance in modern ways not tinged with a slight note of regret?

Let us acknowledge that it is, and with the kindest of intentions here give the thought expression. Fifty years ago "the door was opened," foreign influences poured in like a flood, the best fruit of European civilization dropped full-ripe into the outstretched hand of Japan.

But modern civilization is, in its essence, a continuing progress, its best is always becoming better, and no part or component unit of it can exist independent of the whole. We have found our Japanese friends, so believe some of us, too prone to cut themselves off from the outside influences which have largely made the new Japan, too eager to do everything for themselves, too anxious to feel independent of outside help.

Such tendencies are to be seen in the general exclusion of foreigners from management and superintendence of mines and smelters; in the backwardness of professional Japanese to take an active part in international scientific societies, in their growing disinclination to master English, German and French, and particularly in the general exclusion of foreigners from the faculties of universities and technical schools.

The professional Japanese of the passing generation are much more cosmopolitan in their inclinations than the rising generation; the latter appear in general much narrower in their sympathies and aspirations. It appears, therefore, that the intellectual stimulus of foreign origin is being largely shut out, that the idea of the sufficiency of Japanese for the needs of Japan is gaining ground, and that "the open door" is largely for "beans" rather than for "brains," for commerce rather than for intellect.

In taking this position the Japanese are making a mistake of huge proportions. They fail to take into account the incubus which the mass of their population is to modern progress, the immense drain upon the upper classes in the training of the lower, the great drag on the intellectual resources of the educated Japanese which his environment and daily contact with the people force upon him.

The educated classes of Japan are not alone sufficient to the task before them; they themselves are constituted more than half "old Japan" in feeling, thought and sentiment. They can lead their people, but they still require the external stimulus of intellectual intercourse with foreigners, liberally applied, to prevent discouragement and retrogression.

Perhaps the reflex of a recently successful war has something to do with this overconfidence in the intellectual self-sufficiency of the Japanese for Japan. Whatever the cause, the pendulum has swung too far; the previous policy may have been unnecessarily liberal in utilizing foreign assistance, but the present policy is undoubtedly too stringently excluding it.

Let us hope to see, in the near future, more foreigners managing and superintending mines and smelters, more Japanese active members of international scientific societies, more professional Japanese masters of the English, German and French languages, and more foreigners of high ability upon the faculties of their universities and technical schools.

This is the best that we can wish for Japan, and may it thus maintain its marvelous crescendo of progress.

* * *

Discussions at San Francisco.

To supplement the report of the San Francisco meeting which appeared in the November issue, the following abstracts of some of the discussions will be of interest.

Alaska Coal was the subject of a spirited discussion, led by Mr. H. Foster Bain, editor of the *Mining and Scientific Press*. Mr. Bain urged that some kind of public control of the Alaska coal deposits is desirable, and that the simplest form of control yet found is ownership. However, in his opinion, government ownership of a part of this coal, say one-third, would attain the desired end of preventing individual monopoly of the field. From this reserved land the government itself should mine coal to supply the needs of government vessels or railways. Further, the government should lease coal lands to individuals on liberal terms, for a fixed period, such as 25 years, at a low royalty, the number of such leases being restricted to correspond to the probable market, so as to prevent overproduction and consequent waste.

Dr. R. W. Raymond, secretary-emeritus of the Institute, ridiculed the above proposition as impractical and un-American. He believed in as little government as possible, and that it was puerile to substitute a government monopoly for the dreaded individual monopoly.

The fallacy of Dr. Raymond's protest was, however, cleared up by a member pointing out that "monopoly" by the government, in the interest of the people as a whole, was an entirely different matter from monopoly by private individuals in the

interest of private capital and at the expense of the people as a whole.

One sentence of Dr. Raymond—"You cannot cure national evils"—was so brutally pessimistic and so contrary to facts that his further tirade fell on deaf ears.

The general opinion of those who heard the entire discussion was that while Mr. Bain had not presented an entirely satisfactory solution of the question, yet something, along some such lines, should be done in order to meet a situation whose dangers were entirely evident to everyone except Dr. Raymond.

Gold Dredging in California was handled very thoroughly by Mr. Charles Janin. He gave the history of the development of the dredges, as means of reaching the deep gravels, 40 to 60 ft. below the surface, which could not be otherwise economically treated. The present dredges cost up to $250,000, handling over 20,000 tons of material per day. By this means alone, the output of gold has risen in California from $18,000 in 1898 to over $7,000,000 in 1910.

Statistics of Gold Production in California were given by Mr. Charles G. Yale. He credited the profession of mining engineers with having done more for the state than even the pioneers, for while the latter often wasted more gold in tailings than they extracted, the former were preserving the resources of the country and utilizing them with a minimum of waste and loss. Counting from 1792 to the present, California has produced one-half of the total gold output of the United States.

California Oil was the subject of a paper by Mark L. Requa, which excited great attention. He deprecated the reckless manner in which the oil resources of the state are being exploited, and commended the action of the government in withdrawing oil lands from entry.

With the present rate of overproduction, oil brings at the wells only 30 cents per barrel, and is handled and gathered so carelessly that the loss by seepage, waste, etc., is estimated at 3,000,000 barrels per annum. With oil at a higher price and more restricted production, much of this waste would be avoided. The opening up of more oil lands would have the opposite effect. At double present prices California oil could compete with Alaska coal along the Pacific coast, from Vancouver to Panama, and at present prices from the Arctic circle to Chile.

The question of opening Alaska coal lands to entry is, therefore, one of no present importance, except for the small demand locally in Alaska. He estimated the total oil supply of California at 8,500 millions of barrels, which is about half of the total oil resources of the United States.

Transvaal Gold Production.—The number of companies reporting to the Transvaal Chamber of Mines in September, 1911, was sixty-nine. The total tonnage milled during that period was 2,088,522 tons of ore; 10,307 stamps were in operation with an average duty of 7.91 tons per twenty-four hours. Tube mills in commission numbered 235. The yield for the month was 700,625 fine ounces gold. The tonnage treated, number of stamps and tube mills at work, and total yield, all show marked increase over September, 1910.

The Colorado Scientific Society has inaugurated a movement which is being received with enthusiasm by the various mining and milling districts of the state. A series of practical talks by members of the society will be given at various towns, the subjects and speakers being selected appropriate to the peculiarities of the district. The first of these talks was held at Idaho Springs, where Mr. Charles A. Chase spoke on "Mine Management." At Leadville George W. Schneider spoke on the "Application of Business Principles to Mining and Metallurgy." The latest meetings have been held at Telluride and Ouray, where Mr. A. E. Anderson spoke on "Recent Developments in Explosives," and Mr. C. J. Stahl on "The National Forests and the Prospector." Other meetings will be held at intervals throughout the coming year.

Nomenclature of the Microscopic Constituents of Iron and Steel.

BY HENRY M. HOWE.

Socrates: "But why should we dispute about names when we have realities of such importance to consider?"
Glaucon: "Why, indeed, when any name will do which expresses the thought of the mind with clearness?"
Plato: "The Republic."

An attempt is about to be made to adjust and systematize the nomenclature of the microscopic constituents of iron and steel, and on this account some reflections on this general subject may interest your readers.

When we learnt from Sorby and Osmond that steel was composed of several different microscopic substances, each of which was important for one reason or another, we saw that we needed short names for these substances, so that we might the more easily speak and write about them. Thus during the last quarter of a century a distinct nomenclature has sprung up which has been of great convenience. Unfortunately, usage has not been strictly consistent. A given name, for instance "sorbite," has been used by some in one sense, by others in a slightly different sense. The International Association for Testing Materials has appointed a committee to report on this subject at its coming congress in New York in September, 1912, in order to prevent further confusion by giving the several names clear and consistent definitions. On this committee are specialists from the chief countries in interest, France, Germany, Great Britain, Sweden and the United States.

That uniformity in nomenclature is needed here as it is everywhere else nobody will question. But serious obstacles present themselves, as to which I should like to say a few words. I believe that in the last analysis they are due to misconceptions as to the functions and limitations of language in general, and as to the needs of such a nomenclature in particular. This I will try to show as I explain the general conditions.

When molten steel solidifies it forms what is called a solid solution of iron with carbon, the molecular constitution of which does not concern us here. This is called by most writers "austenite," by others "gamma iron" and by others "mixed crystals." When steel first solidifies the whole of the carbon present is in general contained in this austenite, so that austenite may have any carbon content up to about 1.7 per cent. In cooling slowly this austenite is transferred progressively, in passing down through a certain critical or "transformation" range, into a granitic conglomerate consisting of (1) particles of relatively pure iron, mechanically mixed very intimately with (2) other particles of the definite iron carbide, Fe_3C, in a fixed ratio of about six of iron to one of carbide; and (3) in addition further particles, more or less abundant, of either such iron or such iron carbide, according to the carbon content of the whole. The particles of nearly pure iron are called *ferrite*, those of the iron carbide are called *cementite* and the intimate mixture of these two in the ratio of 6:1 is called *pearlite*.

When cast iron solidifies it forms a like solid solution or austenite, and in addition more or less graphite and cementite. During its slow cooling part of this cementite breaks up into a mixture of graphite and austenite, a change which we need not consider here; but its initial austenite, like that of steel, transforms, in cooling down through the transformation range, into a like granitic conglomerate of pearlite with either ferrite or cementite, according to the carbon content of the austenite itself which thus transforms.

Were this all, though it might be complex enough, it would not be likely to give rise to any serious differences as to nomenclature. Unfortunately for purposes of explanation and nomenclature, this transformation from austenite is a sluggard and a laggard, though in fact its very sluggishness is of inestimable value to mankind. Just as we should be basely ungrateful if we complained of friction, because without friction no man or

beast could stand erect an instant, and every unreinforced masonry structure would at once fall down, so we should be profoundly grateful for the complications which arise through the sluggishness of this transformation, because to this sluggishness we owe the hardening power of steel, and the various ways of bettering it by heat treatment. In the same way we owe the transparency of glass to its sluggishness in transforming into its opaque state of true equilibrium, a sluggishness without which we should be housed without daylight, and should be almost wholly ignorant of chemistry, physics and astronomy.

Thanks to the sluggishness of this transformation, and to the sluggardizing effect of the presence of manganese and nickel, we are able to harden steel, by cooling it too fast to allow the transformation to complete itself, and again to mitigate or "temper" the effects of that hardening, by appropriate heat treatment, which allows the arrested transformation to go on to a definite and predetermined degree. That is the welcome side of the sluggishness, that it increases the value of the metal incalculably; its seamy side is that metallographers will quarrel about the interpretation and the names of these various transition stages between the end conditions, austenite before the transformation begins, and pearlite with ferrite or cementite after it is complete.

What are these transition stages?

When steel is cooled with great rapidity from above the transformation range the transformation is arrested in an early stage recognized by its acicular structure, and under favorable conditions even part of the initial austenite may be retained closely intermixed with this acicular substance, which is called "*martensite*."

If the transformation is allowed to go a little farther, it may yet be brought in either of two fairly well marked stages, troostite and sorbite, or in the less prominently marked stage osmondite. Most of our troubles as to nomenclature concern these three intermediate stages. Looking broadly at these microscopic substances, both end and transition, two of them are definite chemical substances, the pure iron or ferrite, Fe, and the iron carbide, cementite, Fe_3C; one of them, austenite, is a solid solution, the composition of which, like that of salt water or any other solution, varies by infinitesimal degrees from specimen to specimen, and one, pearlite, is a conglomerate or aggregate of ferrite and cementite in definite proportions.

Ferrite, cementite and austenite are phases, in the language of the physical chemist, i.e. any given specimen when in equilibrium is chemically homogeneous. They are like the minerals of nature, and therefore called "metarals." Pearlite is not a phase but a mechanical mixture; it is called an "aggregate" to distinguish it from the metarals proper. It is on a par with the petrographic entities like granite, syenite or diorite, as distinguished from the true minerals of nature with which austenite, ferrite and cementite are analogous.

As to the nature of martensite, sorbite and troostite, we do not know positively, though there is strong reason to believe that sorbite consists chiefly of the final products of the transformation, pearlite with either ferrite or cementite, emulsified, i.e. in so fine a state of subdivision that they cannot be recognized with the strongest magnification. But that does not here concern us. As long as opinions differ, let us recognize that they do differ, and proceed exactly as if we did not surely know. That can do us no harm in framing or defending our nomenclature.

At the last congress of the association a report was presented which included the transition stages, troostite and sorbite, and less prominently osmondite. The framers of that report could not be present to explain or amend it, and in their absence a different scheme was adopted by the congress. This scheme completely suppressed "troostite" and "sorbite," proposing that "osmondite" should replace both of them. This step seemed to me an error, for the cogent reason that each of the two stages, troostite and sorbite, is of an importance, both micrographically and industrially, to justify retaining its already established name. Beyond this, M. Osmond refused categorically, and, in my opinion, justly, to permit this use of his name.

In effect metallographers have gone right on as if that report did not exist. Most of them have continued to write of troostite and sorbite, and some have continued to write of osmondite. Thus to-day the names of the several stages, martensite, troostite, osmondite and sorbite, are used either by most writers or at least by a group too important to be ignored. In addition the names of the end substances, austenite, ferrite, cementite and pearlite, continue in very wide use.

Under these conditions the duty of the committee in charge seems to me perfectly clear. (1) To interpret to readers the meanings of these various terms as used currently so that the reader may understand what the writer means. (2) To erect boundaries and definitions which shall distinguish the several microscopic substances from each other. (3) Where a given name has been used with different and inconsistent meanings, to advise the abandonment of all but the best, if possible securing the acquiescence of those who have used the meanings which it advises abandoning. (4) Where more than one name has been used to represent a given entity, to record, for the benefit of the reader, all those which have been used to an important extent, and to advise the abandonment of all but the best, as before seeking the acquiescence of the parties in interest. (5) To describe the several substances and to explain their genesis and life history, indicating the more reasonable of the underlying theories.

The task of the committee is made more difficult by the attacks from various quarters made against the existing names.

First comes the charge of the physical chemists, flushed with their late victories over nature, and overconfident of the advantages of their point of view. With their minds set on phases, they assume that only phases deserve a name. One eminent metallographer says "it was certainly better that names should not be given to things which could not find a place on the equilibrium diagram." They would erase the names "pearlite," "martensite," "troostite" and "sorbite," and a fortiori "osmondite," on the ground that those things which they represent are not phases. The position is a natural one, and, in view of the intelligence and importance of those who take it, it certainly must be treated with respect.

They lose sight of a very important consideration, that an existing name may properly be retained if it is useful for any purpose. The equilibrium diagram itself is of the greatest value; yet it is not an end in itself, but only a means toward an end, a useful tool to interpret and foretell. Whenever and wherever a substance or a group of substances is of such importance that the convenience of language calls for a specific or generic name for it, such name will arise, and will continue in use, though all who do not happen to need it should thunder protests to Heaven. The simple truth is that, though each of these several names may be needless for the purposes of the equilibrium diagram, they are of use either for metallographic or industrial purposes, or for both.

How useful names may be which do not find any place on the equilibrium diagram may be understood by seeing how useful, in the closely allied subject of petrography, names are which find no place on the geophysical equilibrium diagram. Clay, shale, schist, etc., are not phases; but they are names of great usefulness, simply because they stand for classes each of which is convenient for purposes of speech.

To this the reply is made that, even if such classes may be named, yet they must not be given the suffix "ite."

But, first, there is nothing peculiarly sacrosanct about this suffix. It is true that most of the modern names of true minerals end in ite; but it is equally true that many of the most typical and best known mineral names do not, quartz, idocrase, tourmaline, psilomelane, galena, and so on by the dozen.

But "ite" is not distinctive of mineralogical rather than

petrographic entities. Granite, trachyte, dolorite, diorite, diabase, hyperite, aphanite, syenite, pegmatite, quartzite, steatite, ophiolite, etc., are not phases, but either conglomerates or special states of aggregation. Most of the petrographic "ites" are not phases.

The reason why these mineral species and these rock species have received specific names is that convenience of speech demands it. It is convenient to have one single name which shall cover all granite, etc., etc.

Then come others who would deny to martensite, sorbite and troostite the suffix "ite" on the ground that they are of variable composition, and others would deny this suffix to even austenite on this same ground. They know not what they say. The suffix "ite" is expressly for the purpose, among others, of naming substances of widely varying composition. It is, indeed, widely and properly used for substances of composition varying incomparably more than the composition of austenite, martensite, troostite or sorbite varies.

Let us take austenite as an example. It is gamma iron containing dissolved carbon which cannot under any conditions so far as we now know exceed 1.7 per cent. It is not necessary to go to the rock substances, granite, syenite, etc., to find such variations in composition, for the true minerals of nature vary incomparably more. What mineral is there which never has one of its essential components, elements or radicals, replaced by another to an extent greater than 1.7 per cent? I take almost at random eight minerals from Dana, with the variations in percentage of one of their essential components.

Chromite:	Ferrous oxide	varies	from	19	to	39	per cent
Spinell:	Magnesia	"	"	10	"	27	" "
Magnetite:	Ferrous oxide	"	"	12	"	29	" "
Dolomite:	Magnesium carbonate	"	"	10	"	68	" "
Ankerite:	" "	"	"	12	"	36	" "
Smithsonite:	Zinc "	"	"	40	"	98	" "
Corundum:	Alumina	"	"	50	"	93	" "
Menaccanite:	Ferric oxide	"	"	2.66	"	94	" "
	Titanium "	"	"	3.55	"	59	" "
	Ferrous plus ferric oxide	"	"	40	"	96	" "

It is quite permissible to regard austenite as essentially gamma iron which may contain carbon dissolved in it. From this point of view the variations in its carbon content are insignificant compared with the variations in composition of the minerals of nature. If austenite must have its queue cut off because its composition is not fixed, which of the names of the minerals of nature shall not deserve like mutilation?

Beyond this, we have no reason to believe that the physical properties of this austenite vary more, with its varying carbon content, than the properties of certain rock species, such as granite, or even mineral species such as obsidian. It is true that the products of the transformation of that austenite arrested by rapid cooling do vary, from pure ferrite in the case of nearly carbonless austenite, through martensite of varying carbon contents, to the mixture of austenite plus martensite present in rapidly cooled steel of 1.50 per cent of carbon. But that is not to the point. With the exception of its fusibility, the properties of the austenite itself, as it exists above the transformation range, have not been shown to vary greatly with its carbon content. Certainly there is no evidence of variations of a kind that make a single specific name like austenite inapplicable to the whole range of the solid solution above the transformation range.

Why are minerals called by their mineralogical names rather than by chemical names? Why do we speak of "corundum" rather than of "alumina"? Partly because corundum, though typically consisting of alumina, is in fact rarely identical with alumina. Indeed it may contain only 50 per cent of alumina. An additional and very strong reason is that the mineralogical name implies nothing either as to chemical composition or molecular constitution.

Here we see at once why the pioneers in metallography selected mineralogical names, (1) because we were and are uncertain or at least in dispute as to the constitution of these microscopic substances, so that a mineralogical name, through its implying nothing on these heads, is appropriate; (2) because the composition of these various microscopic substances varies greatly, so that a chemical name, with its implication of fixed composition, would be inappropriate. Thus the very reasons which are raised for denying the suffix "ite" to austenite, martensite, troostite, and sorbite, that their composition varies and that their constitution is in dispute, are really reasons why their names should be mineralogical. To me these reasons seem amply sufficient.

Next come certain persons of importance who would remove the suffix "ite" from martensite, etc., on the ground that martensite is only a special structure and not a distinct entity. But if that were true it would be no reason at all. Between quartz, quartzite, itacolumite and novaculite the difference is essentially one of structure. So with the varieties of calcite. The names iceland spar, preunnerite, satin spar, aprite, travertine, stalacite, and stalagmite are based essentially on structural differences. So between chalcedony, agate, onyx, flint, hornstone, and jasper the difference is chiefly one of structure. Remember that these last names are exactly on a par with those ending in ite, and the fact that they do not is due solely to their antedating the time when the suffix "ite" was adopted conventionally. So with granite, granulite, pegmatite; their essential difference is one of structure.

A third class, also containing men of importance, objects strenuously to the name osmondite. With reluctance they will allow troostite and sorbite to remain, provided osmondite is stricken out. Their objection is not clearly formulated; but it seems to consist of two parts; first, osmondite is not shown to be a phase; second, it is not necessary to have a name for a stage intermediate between troostite and sorbite. Both these objections they hold against troostite and sorbite also; but they waive them. Let us consider them as applied to all three names.

The first objection we have already seen to be insufficient because it applies with equal force to many true mineralogical names, and to petrographic ones, whether ending in ite or not, so that the enforcement of this principle would be unwise.

Turning now to the question whether these names are needed, it really resolves itself into two, one for the writer and one for the lexicographer, in which latter class the present committee belongs. It is for each writer to decide whether he does or does not need names with nice shades of meaning, such as osmondite and troosto-sorbite. Those who look only at the general outlines and not at the details have no right to forbid the workers in detail from having and using words fitting their work; nor have those whose needs are satisfied by the three primary colors a right to forbid painters, dyers, weavers, and others from naming the many shades with which they are concerned. To the dweller in a perfectly flat country the names hill and plain may suffice; yet those who live in the foothills may well need many more names, rise, swell, mound, hummock, hillock, mountain, crag, butte, canyon, valley, vale, dale, swale, gulley, etc. Stanley told me that the language of the African forest dwellers was incomporably richer than English in names for varieties of growth, forest, jungle, etc., simply because, savages though they were, their environment gave an importance to differences which seem to us unimportant and called for distinctions which to us are superfluous. The smith needs a few words which are needless to the poet, and the poet has thousands which the smith knows not. It would be unfair to either, if it were not futile, to forbid him to use those which are needless to the other.

So much for the aspect of the matter to the writer and speaker. The committee is charged with a lexicographic duty, to teach the reader what the writer means when he uses a certain word so as to help the reader to understand the writer. If the committee goes beyond and explains current theories,

describes the varieties and their genesis, that does not relieve it of its first duty, to interpret to the reader the unknown word. It may properly say that such and such a word is rare, or obsolescent, or misleading, or ambiguous, and it may even remonstrate against its use. But when it has done that it has exhausted its corrective powers. It has no moral right to say "We will indeed interpret to the reader the words which please us, but we refuse to interpret those of which we disapprove. Let the reader who meets such words remain in ignorance as to the writer's meaning or let him find out as best he may; we owe him nothing, and we refuse to enlighten him." Yet that is exactly the attitude that many would have us adopt.

That proposition seems to me unreasonable. In my opinion, the chief criterion for deciding whether to include any given name in the list is the probable utility of that inclusion. The writer of an alleged unabridged dictionary is under an implied contract with the buyer to interpret to him not only language in its purity, but also colloquial, obsolescent, ill-formed, misleading and otherwise objectionable words, basing his decision as to including or excluding them chiefly on the consideration whether their inclusion will help enough readers and often enough to justify that inclusion. So, in accepting our commission we assumed a like duty toward readers and we should be governed by a like consideration.

Needless words will die a natural death; needed ones we cannot kill. The good we might do in hastening the death of the moribund by omitting them from this report is less than the good we do by teaching their meaning to those who will meet them in ante-mortem print. These readers have rights. We serve no class, but the whole.

Here we must remember that language, though nothing but man's tool, in the nature of the case must be a universal tool with all the defects common to universal tools. We must all use our mother tongue; and if confusion occurs when the poet uses words unknown to the smith and vice versa that is a necessary consequence of the universality of language.

Applying this principle to the present case, I cannot understand how there can be a shadow of a doubt about including troostite and sorbite. Not only do they occur very frequently in the writings of the best metallographists, but they represent substances of real importance. For certain industrial uses the troostitic state is invaluable; for others the sorbitic state is indispensable. The micrographer often needs "troostite" and "sorbite" to indicate what he sees. If they have already become convenient in this early stage of the development of heat treatment, how much more useful will they be hereafter?

When we come to osmondite the matter is less simple, because a smaller proportion of competent writers either use the name or believe that this additional stage in the transformation is of such importance as to justify giving it a special name.

Though I certainly cannot go so far as to say that this name is strongly needed, yet I am perfectly confident that the plain duty of the committee is to include it in its list. If I have no use for the color name "lavender," finding "violet," "lilac" and "slate color" quite sufficient, who am I that I shall forbid my neighbor to say "lavender" if he sees fit? Certainly the intermediate stages of the transformation have a present and prospective industrial importance so great that, when a writer of such weight and standing as the proposer of this name assures us that he finds need of further subdividing these stages and recognizing the osmonditic stage between troostite and sorbite, we can readily understand that he may be right, even if we personally have not felt the need of naming this stage. In view of the importance of his writings and of those of his followers, it is likely to be a convenience to a large number of readers to have the meaning of "osmondite" set forth in the committee's report. The help that would thus be given those readers certainly seems greater than any possible harm that can be done by recording and explaining the meaning of "osmondite." Hence it seems to be a duty to those readers to include and describe osmondite.

Thus the duty of the committee seems clear without certain other reasons of a different class which I am reluctant to invoke. For, though to some they seem of great force, indeed to be part of the general policy of "peace on earth, goodwill towards men," yet they seem opportunistic to others, and unfortunately these others are just the ones hardest to convince.

For instance, language is a convention, and the wishes as such of the parties in interest should be considered.

The propriety of its names is not in general dependent on any theory, now it is liable to be affected by later discoveries. The names are matters chiefly of taste and convenience, adopted as a matter of convenience and mutual agreement by those who have to use them.

Moreover, a committee should take care that it does not exercise its power arbitrarily, and for the gratification of the feelings of a majority of its members, remembering that, whereas little if any harm can be done by including a word which seems to us distasteful or needless, because nobody is forced thereby to use it; yet we may be mistaken in thinking it needless, and if the word is really needed, harm would be done by excluding it, not only through leaving readers occasionally in ignorance, but through defeating the whole plan for a uniform standard nomenclature.

Psychological considerations cannot be ignored. Personal and racial antagonisms, the odium metallurgicum, and other considerations, are here, and we need not shut our eyes to them. They should lead us to discount all pleas for excluding names, unless supported by cogent reasons.

Finally, as regards the nomenclature as a whole, the convenience of readers requires that the meanings of words should be changed as little as possible unless for strong reasons. In the present case this consideration is reinforced by two further ones, first that our whole knowledge of the life history of the substances to which these names apply is now most incomplete, and a great increase of it is to be looked for; and second that mineralogical nomenclature itself is not unlikely to receive a methodical readjustment. The time of that readjustment would be the proper time for readjusting metallographic nomenclature, so that the two may correspond.

For both these reasons it would seem wise to disturb the existing nomenclature as little as possible, and to confine our work to adjusting, rectifying and perfecting it, so that it may serve its present purpose.

Most of the present names have been in general use for nearly a quarter of a century; they are tools to which our hands are wonted. To attempt to wrench them from us and forbid their use would be as unwise as it would be futile. That sort of prohibition certainly would not prohibit.

Columbia University,
New York City.

Continuous Carbon Dioxide and Temperature Recorder.—In the article by Mr. Ed. A. Uehling, published in our December issue, page 656, there is a misprint in formula (5) on page. 657. The second term in the brackets should be $3.84Ha$ instead of $0.88Ha$.

Wage and Bonus Systems.—The Emerson Company, 30 Church Street, New York City, has just issued an interesting pamphlet, entitled "A Comparative Study of Wage and Bonus System." After a general introduction on the systems of paying for work on a basis of status or on a basis of contract or on a basis of individual effort, the characteristic features of bonus systems are explained. By means of simple diagrams the following systems of compensation are very clearly represented: time rate, piece rate, Halsey one-third bonus system, Halsey two-third bonus system, Rowan premium system, Taylor differential piece-rate system, Gantt bonus system and Emerson bonus system. The latter is described in some detail. The little pamphlet is well worth the attention of employers of labor.

Cyaniding by Continuous Decantation in Clear County, Colorado.

BY H. C. PARMELEE.

In the oldest mining district in Colorado, where the stamp, the amalgamating plate and the concentrating table have held sway for almost fifty years, some of the latest ideas in cyanide practice are in successful operation. One is not accustomed to think of cyanidation in connection with the silicious sulphide ores of Clear Creek county, although at intervals during the past five years several extensive experiments were carried out on a laboratory scale which indicated the amenability of many of those ores to that process. Higher recovery and greater profit were indicated in these tests as compared with former practice, but for one reason or another no commercial operations resulted.

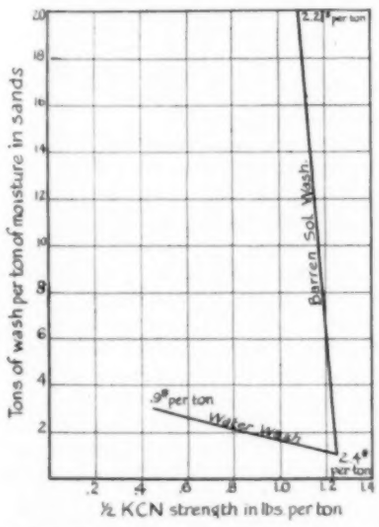

Tons of wash per ton of moisture in sands
2.2# per ton
Barren Sol. Wash
.9# per ton
Water Wash
2.4# per ton
½ KCN strength in lbs. per ton

FIG. 1. DECREASE IN KCN IN SANDS COMPARED ON WASHING SANDS WITH BARREN SOLUTION AND WITH WATER.

There are many difficulties to be overcome in an old district which do not obtain in a new one. Conservatism and skepticism combine to retard progress, making it difficult to procure money; and when these difficulties have been overcome, economy suggests the use of old mills and equipment as far as possible. These conditions limit the metallurgist and work a hardship on the industry in general for although the results show an improvement they are not the best that it would be possible to obtain with new, well-designed plants and modern equipment.

It has remained for the Hudson Reduction Company, Idaho Springs, Col., to treat successfully by amalgamation, concentration and cyanidation, the varied ores produced in that district. Experimental work was commenced about a year ago under the direction of Arthur H. Roller, manager, and Justin H. Haynes, metallurgist, and the present plant is the result of their preseverance under many adverse circumstances. An old twenty-stamp amalgamation-concentration mill has been remodeled to include cyanidation by continuous decantation in Dorr slime thickeners. Filtration is eliminated, excepting the use of clarifying leaves preceding precipitation of the cyanide solution. Warm solutions are used throughout the plant, being maintained at about 30° C. at the zinc boxes, and electrolytic oxygen is being experimented with as a means of keeping the solutions well oxidized.

The mill is operated on a custom basis, the ores being bought on a schedule based on their gross valuation. The ores are mainly silicious sulphides of iron, although some contain lead and copper, the latter metal averaging about 1 per cent in the mill heading. The ratio of silver to gold averages about 15:1. The great variety of ores offered for treatment, and the present lack of bedding facilities or bin room, make it impossible to depend on a uniform feed to the mill and demand close attention to the solutions. The average gross value of the ore is about $10 per ton, but the ratio of concentration in the different ores varies widely, as also does the presence of cyanicides and the percentage of slime formed. Notwithstanding these conditions, the average recovery is 88 per cent. which is probably 25 per cent higher than that of old practice in the district.

The addition of the cyanide process has not greatly altered the former percentages of recovery by amalgamation and concentration, but has converted a former tailing loss into additional saving. On $10 ore about $1 per ton, or 10 per cent, is recovered on the amalgamating plates. The percentage recovery by cyanide varies with the concentration ratio and the quantity of slime formed. With an ore of high concentration ratio perhaps 30 per cent of the total value is recovered in the concentrate and 50 per cent by cyanide, while the figures are practically reversed on an ore of low concentration ratio. Whether amalgamation and concentration would be retained as features of a new plant would have to be decided by experiment and would depend to a great extent on the available ores. Coarse gold would suggest amalgamation unless fine grinding could eliminate that step. Ores containing galena would require concentration. It is possible therefore that amalgamation might be dispensed with, although improbable that concentration could be entirely eliminated. At the Hudson mill the solubility of much of the gold and silver in cyanide solution is quite marked, about $1.50 in value going into solution in the batteries.

Sampling.

The sampling is well done, although at greater expense than would be necessary in a better arranged plant. The first reduction in size is to 1½ in., this product being elevated to a Vezin sampler where the first cut of 20 per cent is made, the reject going to the battery bins. The 20 per cent sample is then crushed to ¾-in. size, and again cut by a second Vezin machine giving 20 per cent, or 4 per cent of the entire lot. The final cut is then crushed to ⅜-in. size for coning and quartering, this operation resulting in a sample of 15 lb., which is ground and split on Jones riffles to about 1 lb. The final sample is then ground through 120-mesh screen.

Milling.

In these notes reference is made to the accompanying flowsheet and plates which represent graphically some details of treatment. The capacity of the plant has been increased from 60 tons to 88 tons per day since the flowsheet was drawn, but the figures given are proportionately correct. The plates, also, may not correspond exactly with the average figures given in the notes, for the conditions vary greatly from day to day, and the curves may represent conditions during a short period of time, while the figures given in the notes are averages over longer periods.

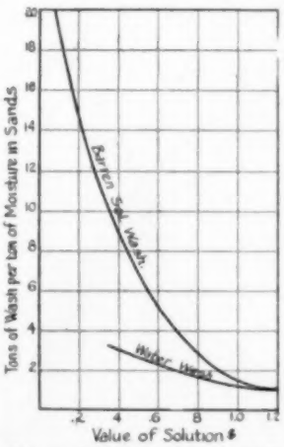


FIG. 2. DECREASE IN GOLD CONTENT IN SAND SOLUTION COMPARED ON WASHING SANDS WITH THE BARREN SOLUTION AND WITH WATER.

The ore is crushed in cyanide solution by twenty 850-lb. stamps, dropping ninety times per minute, with a height of drop of 6 in. Ton-cap screens are used equivalent to 12-mesh, giving a stamp duty of 4.4. tons per stamp per 24 hours. The plate area is 52 in. wide by 12 ft. long for each set of five stamps, and is arranged in three drops. Practically all of the amalgamation occurs on the uppermost section.

The pulp is immediately concentrated without classification on four tables as shown in the flowsheet. Four products are drawn from each table and combined. *Concentrate* is elevated to an Akins's classifier (not shown) and dewatered, for ship-

ment to the smelting plant. Experiment has shown that the concentrate can be cyanided separately from the gangue, and further tests will be made looking toward cyanidation of combined concentrate and slime. In this event all concentrates and middlings from tables would be reground and combined with the slime.

Middling from the four tables is then at once retreated on a fifth table making concentrate and coarse and fine tailing. The concentrate and fine tailing are combined with similar products from the first four tables, while the coarse tailing is reground in a 5-ft. grinding pan, 85 r.p.m. and combined with the slime. *Sand tailing* is pumped to an Akins classifier for separation of contained slime, and then washed by a large volume of barren solution to a dewatering table, from which it is discharged to waste. *Slime tailing* is the product from which the principal cyanide extraction is made. After being thickened in a Callow cone it passes by continuous decantation through the agitator and three Dorr continuous slime thickeners and thence to waste.

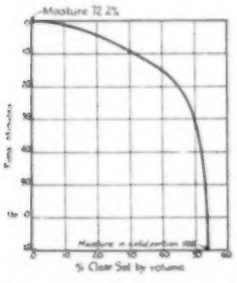


FIG. 3. SETTLEMENT OF SLIMES.

Sand Treatment.

It will be observed that the sand tailing is run to waste without cyanide treatment other than that received in its course through the batteries and on the tables. Under present conditions, at least, this is the best disposition to make of it. In fact, if the table feed were properly classified, and sufficient clear solution were available for table wash water, it is probable that the sand tailing would be so low in grade that it would not repay further treatment.

At present, as will be shown later, a part of the table wash water is slimy solution, and this fact, together with the lack of classification, allows the sand to carry more or less slime which largely gives it value.

From the Akins's classifier the sand is sluiced to the dewatering table with barren solution, used at the rate of 20 tons of barren solution per ton of solution in the sand. This washes the soluble gold and cyanide from the sand quite effectively. Experiments were made looking toward the use of water as a sand wash, but on account of the limited quantity which could be used without greatly increasing the mill solutions, it was found ineffective. The comparison of washing with barren solution and with water is show in Plates I and II. Water was more effective in washing out cyanide, but barren solution removed a greater value in gold and silver.

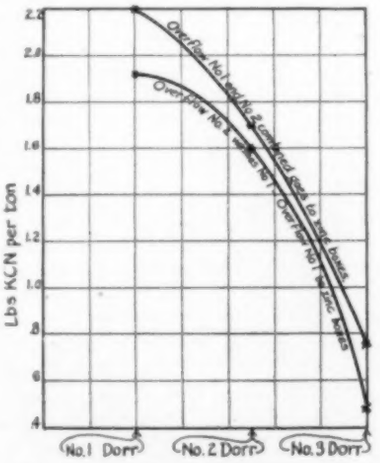


FIG. 4. DECREASE OF DISSOLVED CYANIDE IN DORR THICKENERS UNDER TWO SYSTEMS OF CONTINUOUS DECANTATION.

As used at the Hudson mill a concentrating table is very effective in dewatering sand tailing, an old Card table being used for this purpose. The discharge end is elevated 2 in. above the feed and a cleat nailed along the former to form a bank over which the tailing must discharge. The side pitch of the table is regulated according to the load so that no sand passes over the side with the slimy solution. The net result is that the tailing banks at the discharge end of the table and gradually passes over with an average of 14 per cent moisture. The most efficient dewatering done on the table has been 9 per cent moisture, and the poorest, 22 per cent; but a general average of 14 per cent has been maintained easily, at a capacity of 60 tons of sand in twenty-four hours. The sand tailing carries dissolved gold and silver to the value of $0.012 per ton, and has a gross value of $0.32 per ton.

Continuous Slime Treatment.

The following screen analyses, for one of which assay values also are given, show the grade of material treated by continuous decantation .

Mesh	%	Mesh	%	Value per ton.
+ 80	7.3	+ 80	8.7	$1.28
+ 100	5.9	+ 100	2.2	1.87
+ 150	2.4	+ 150	2.2	0.46
+ 200	3.8	+ 200	4.3	1.05
— 200	80.6	— 200	82.6	2.95

The high value of the minus 200-mesh product is believed to indicate the presence of brittle gold, and the fact that it remains undissolved at this stage of the treatment suggests that it may exist as a difficultly soluble alloy, especially in view of the fact that some gold dissolves so rapidly in the batteries. Plate III shows the results of a laboratory experiment on the settlement of slime from one of the ores treated.

The agitator is a cone-bottom tank 8 ft. in diameter and 24 ft. high, conforming practically to the proportions of a Pachuca tank. Air agitation may be used later, but at present the pulp is drawn from the bottom into a centrifugal pump and discharged downward on to a circular apron in the top of the tank. Thickened pulp from the Callow cone enters the agitator continuously and the discharge from the agitator to No. 1 Dorr thickener also is continuous. The discharge is delivered to the thickener by a launder which cuts a part of the stream flowing over the circular apron. This insures the transfer of a uniform sample of the agitator pulp into the decantation system. The quantity of pulp discharged from the agitator is regulated by a float rigidly connected with the pipe discharging on to the apron. As the float rises or falls it turns the mouth of the pipe slightly toward or away from the launder leading to the thickener, thereby increasing or decreasing the quantity passing out of the agitator.

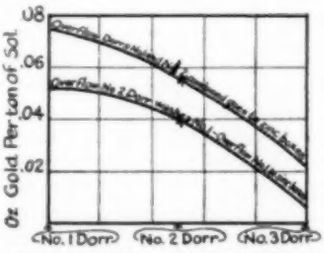
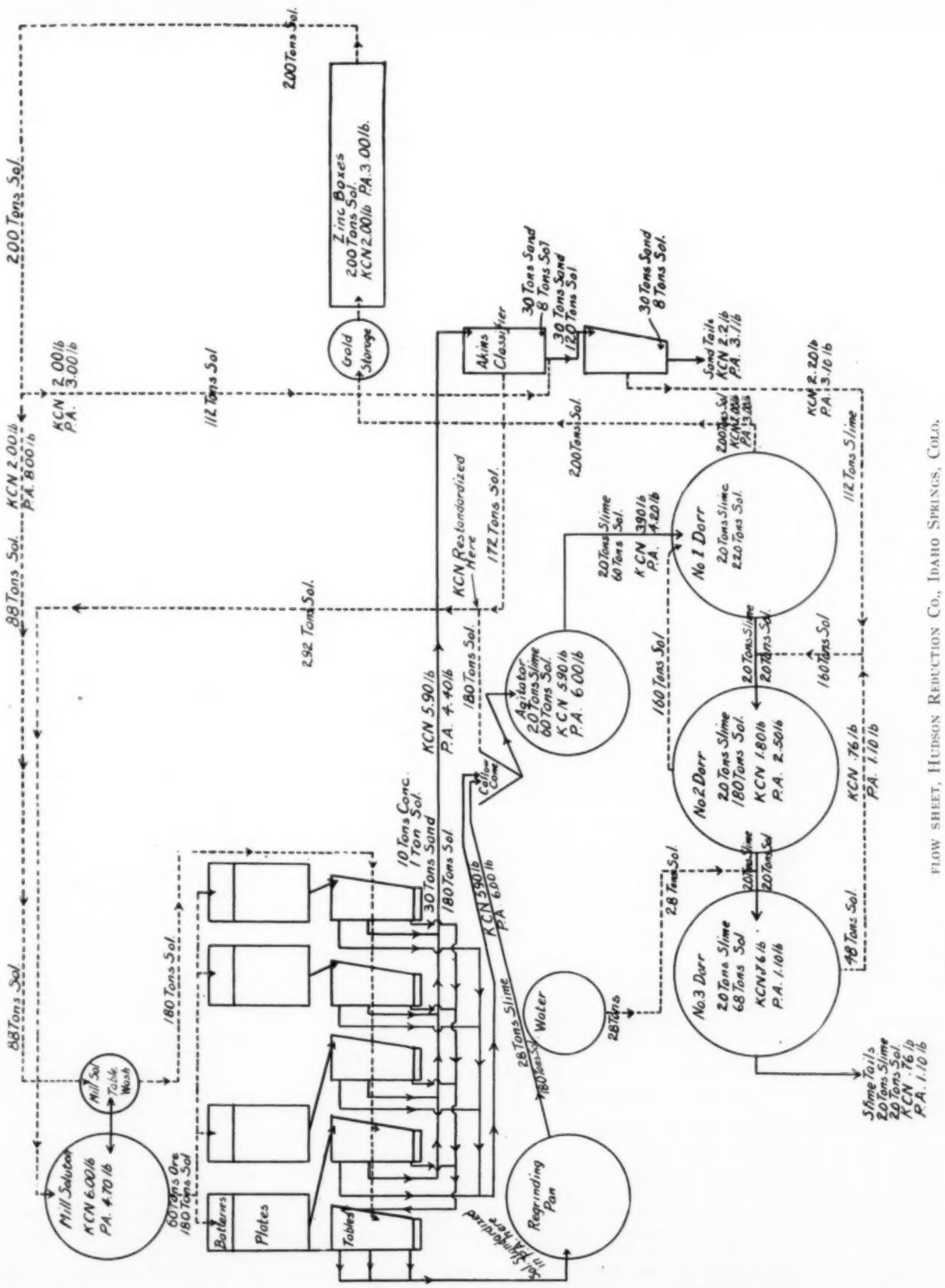


FIG. 5. DECREASE OF DISSOLVED GOLD IN DORR THICKENERS UNDER TWO SYSTEMS OF CONTINUOUS DECANTATION.

The Dorr thickeners are each 16 ft. in diameter and 10 ft. deep, being of these dimensions in order to get them into the space available. All are equipped with spring drive gear. Nos. 1 and 2 make one revolution in ten minutes, and No. 3 one revolution in eleven minutes. The quantity of thickened underflow is regulated by the use of nozzles ranging from 5/16 to 9/16 in. in diameter. Ordinarily the tanks show clear solution to a depth of 4 ft. In the flowsheet twenty tons of slime is indicated as the usual load, but the machines occasionally carry from 50 to 60 tons of slime when certain ores are treated.

A difficulty encountered in the treatment of the ores received at this mill is the foaming of the solutions at various points in the plant, especially where the pulp flows into tanks, such as the Callow cone and the Dorr thickeners. This foaming becomes excessive in treating ore from a loose vein where a great deal of timbering is required, and in consequence of which an unusual amount of decayed wood becomes mixed with the ore. The foam is troublesome because it cannot be confined at any one place in the mill. In order to control its formation on the Callow cone and the Dorr tanks, a specially constructed feed box was placed on each machine. This consists simply of



FLOW SHEET, HUDSON REDUCTION CO., IDAHO SPRINGS, COLO.

an air-tight box extending 2 ft. below the surface of the solution, into which the feed pipe discharges and from which the foam formed therein may be continually siphoned through a pipe running to the sump from which the next thickener receives its feed. At the Hudson mill these boxes are made of wood and have not given perfect satisfaction on account of leakage. A better mode of construction would be to use a metal cylinder closed at one end, save for flanges to which the feed and siphon pipes could be attached.

The continuous counter-current flow of slime and solution through the Dorr thickeners is shown in the flowsheet. In general the slime flows from the agitator to No. 1, to No. 2, to

No. 3, being diluted and washed by solutions passing in the opposite direction. The clear overflow from No. 1 is precipitated after passing through ordinary clarifying leaves and the thickened underflow from No. 3 is discharged as slime tailing. Clear hot water is added as wash for the slime entering No 3

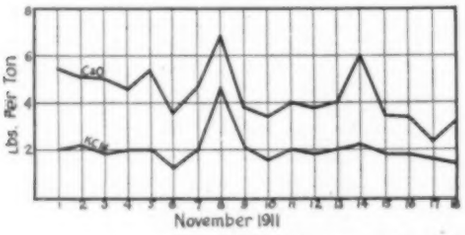


FIG. 6. DAILY CONSUMPTION OF KCN AND CaO PER TON OF ORE, FOR FIRST EIGHTEEN DAYS OF NOVEMBER, 1911.

in sufficient quantity to provide for the solution lost in sand and slime tailings, thereby keeping the quantity of mill solution constant. The quantity of hot-water wash is regulated by a feather valve actuated by a float in the mill solution tank.

Prior to the adoption of the present flowsheet, in which the clear overflow of No. 2 Dorr is used to wash the pulp flowing into No. 1, the experiment was made of precipitating the overflow from both No. 2 and No. 1, instead of from No. 1 as shown. Plates IV and V show graphically the fall in cyanide and gold solution in the three machines under each system and clearly indicate the superiority of the present method.

The slime tailing has a gross value of $1.10 per ton, of which $0.12 per ton is dissolved gold and silver. Present conditions indicate the need of larger agitating capacity, inasmuch as gold and silver continue to go into solution even in No. 3 Dorr thickener from which the tailing is discharged. The value of the general mill tailing, slime and sand combined, is $0.78 per ton.

A study of the flowsheet will apparently show two cycles of mill solution. This is true only in a measure as they are connected at the mill solution tanks. One of the solutions is clear and the other slimy, this arrangement being necessary on account of lack of settling capacity to clarify all the solution. The cycle of clear solution is from No. 1 Dorr to the zinc boxes from which barren solution is used as wash water for the concentrating tables and the sand tailing, whence it ultimately flows back to the decantation system. The slimy solution is used in the batteries, also to a certain extent as part of the table wash water. From the overflow of the Callow cone and the Akins sand classifier the slimy solution is returned to the battery storage tank. When additional settling capacity is provided all the mill solutions will be clear, and an additional zinc box will be provided for precipitation of the present slimy solution, which is really too rich to be wasted.

Precipitation.

The zinc box has five compartments, each 30 in. x 30 in. x 30 in., giving a total capacity of 78.125 cu. ft. of zinc. Two hundred tons of solution are precipitated in twenty-four hours, being 2.56 tons of solution per cubic foot of zinc, or 3 tons per ton of ore treated. The boxes are constructed with 4-in. partitions and a 4-in. space beneath the ¼-in. mesh screen supporting the zinc. The value of the zinc box heading is about $1 per ton and of the tailing about 2 to 3 cents per ton. The temperature of the solution is maintained at about 30° C. at the zinc box, as it has been found that both solution and precipitation are more effective at this temperature than at the ordinary temperature of 15° C. A clean-up is made once a week, the precipitate being dried and screened through a ¼-in. mesh screen and shipped to the smelter refinery. Payment is made at the rate of 98 per cent of the silver, $20.15 per ounce for gold, with no treatment charge and no limit on the quantity of short zinc present.

The consumption of cyanide and lime varies from day to day with the different ores treated, as is shown in Plate VI, where the record is given for the first eighteen days of November, 1911. The average consumption of KCN is 2 lb. per ton of ore; CaO, 3 lb. per ton; Zn, 0.8 lb. per ton, and PbO, 0.5 lb. per ton. The percentage alkalinity of the solution is standardized at the grinding pan, where the litharge also is added. The cyanide strength is standardized in a sump just preceding the mill solution storage.

In spite of all efforts made to keep the solution oxidized it carries reducing agents in considerable quantity. Even with the use of litharge in quantities of 1 lb. per ton of ore, the solution is not in desirable condition. Ten cubic centimeters of

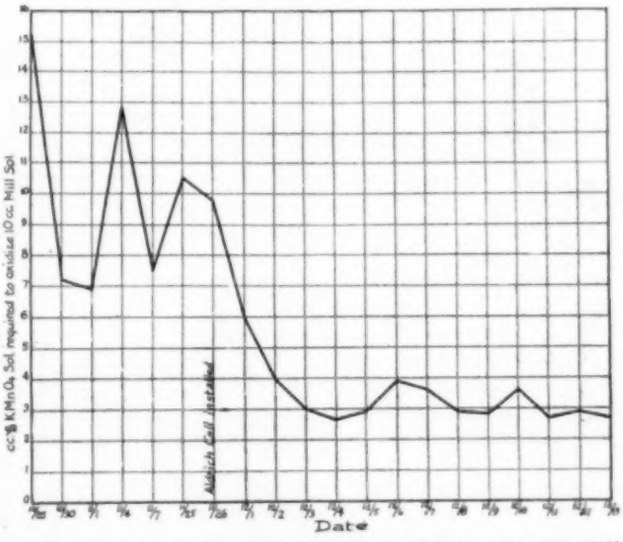


FIG 7. EFFECT OF ELECTROLYTIC OXYGEN ON KEEPING CYANIDE SOLUTIONS FREE FROM REDUCING AGENTS.

solution titrated boiling hot with tenth normal $KMnO_4$ solution requires from 6 to 15 c.c. to produce a permanent pink color. When these quantities of permanganate are plotted from day to day the reducing agents are seen to fluctuate widely. Mr. Haynes has recently started the experiment of using electrolytic oxygen to improve the solutions, following the suggestion of Mr. T. H. Aldrich, Jr., in a paper read before the San Francisco meeting of the A. I. M. E. The results obtained over the short time the experiment has been in effect are very encouraging, the quantity of permanganate required to give a permanent pink color to 10 c.c. mill solution having fallen to 2 c.c., as compared with 6 to 15 c.c. above, and the mill tailing also has fallen in value. Since the installation of this idea the permanganate consumption has fallen gradually and the plotting shows a steadily falling curve instead of the ragged line formerly obtained. Plate VII shows wide fluctuations in the reducing agents prior to the use of electrolytic oxygen and the steady fall and uniformly small quantity present after its adoption.

The electrolytic oxygen cell has been placed in the fifth compartment of the zinc box, where the barren solution will flow through it on its return to the mill. The cell is made up of 111 lead plates, 3.5 by 3 in. in area, alternate plates being connected to one pole of the current. The whole is contained in a hard-rubber box. The current used is 30 amp at 110 volts. If, as appears probable, this device will give highly oxidizing solutions, it may prove to be the solution of the problem of cyaniding concentrates in which reducing agents usually are present in large quantities.

Power and Labor.

The following tabulation gives the power required in the mill, the figures being taken from tests made when electricity

superseded steam. Two motors are in use, one 75-hp motor for the mill and one 15-hp motor for the sampler.

Machine.	Kw	Hp
Sampler	10.0	13.40
20 stamps	22.8	30.55
5 tables	1.6	2.14
Centrifugal pumps	18.4	24.66
44" Frenier pump, 12.5° lift, 50 tons sand, and 300 tons solution.	1.6	2.14
Dewatering table	1.2	1.61
Total	55.6	74.50

Three Dorr thickness and two Akin's classifiers not included.

In the mill two men are employed on each of three eight-hour shifts in addition to the foreman. On the day shift there is an extra man for repair work. In the sampler there are four men working one nine-hour shift.

Summary.

Average gross value of ore	$10.00 per ton
" " " " sand tailing	$0.32 per ton
" " " " slime tailing	$1.10 per ton
" " " " general tailing	$0.78 per ton
" percentage recovery	88%
Consumption of KCN	2 lb per ton
" " CaO	3 " " "
" " Zn	0.8 " " "
" " Pbo	0.5 " " "
Average value of zinc box heading	$1.00 per ton
" " " " " tailing	2 to 3 cents per ton
Tons solution precipitated per cu. ft. zinc	2.56
" " " " ton of ore	3.

The proportions of solution to ore and the percentage alkalinity (P A) and cyanide strength are given at various points on the flowsheet (page 29).

All things considered the work at the Hudson mill is most commendable. The management has proceeded carefully, making many investigations to prove the wisdom of each feature before adopting it, and the thoroughness of the experimental work is a warrant of the ultimate success of the methods employed.

The entire Clear Creek district must be indebted to those in charge of the work for pointing the way to better metallurgy and for their courtesy in making public the details here given. Perhaps further investigation will show that cyanidation has a wider application in the district than is now known and the remodeled Hudson mill may prove to be only the first of other cyanide plants to add to the recovery of the precious metals from Clear Creek County ores.

Calculations of Vacuum Pans.

By B. VIOLA, M. E.,
Member American Society of Mechanical Engineers.

In the evaporation of liquids the main problem is: how can we evaporate the greatest quantity with the least heating surface? Naturally the selection of the most economical medium of a heating surface is the most important matter of this subject. By describing the different evaporators and vacuum pans (this journal, Vol. IX. pp. 206, 251, 306, 349, 389) I have demonstrated the most practical and efficient heating surfaces, and now I wish to describe briefly how to figure the heating surface for the evaporation of given quantities of liquors.

As already mentioned, the materials chiefly employed for heating surfaces are copper, iron, and lead, or in combination copper with lead lining. Lately enameled iron is also used with good success.

Before we go further it is very important to give the heat transmission capacities of these materials. From the experiments of Molier, Ser, Joule, Hagemann, Jelinek, etc., Hausbrand reached the conclusion that the empirical formula

$$K_v = \frac{1900}{\sqrt{dl}} \qquad (1)$$

expresses the coefficient of transmission of heat between steam and boiling water in calories per square meter and hour and per degree C. temperature difference, using cylindrical copper pipes for circulating the steam inside and the water to be evaporated outside, where d is the diameter of the heating pipe in meters and l the length in meters.

This transmission coefficient can be taken for wrought-iron pipes 0.75 time the above value (1) for copper pipes, for cast-iron pipes 0.5 time value (1) and for lead pipes 0.45 time value (1). In practice, however, it is advisable to use only two-thirds of these values. If the solution has a strength of 10 to 25 per cent, this cofficient has to be reduced further by 20 or 30 per cent.

Single Effect.

Now, in order to find how much water can be evaporated by a certain heating surface per minute, I will commence with a single effect, 2 atmospheres absolute pressure, corresponding to a temperature of the heating steam of 120° C.

We suppose that a certain liquor enters the effect at a temperature of 45° C. from the juice heater and by the influence of vacuum pump and condenser we get an absolute pressure of 149 mm. corresponding to a vacuum of 611 mm. (24 in.) and a boiling point of 60° C.

Now we call:

t_0 = temperature of the liquor entering into the evaporator.

t_1 = boiling point of this liquor in the evaporator.

t_2 = temperature of the heating steam.

C = heat transmission coefficient, in calories per square meter per minute and 1° C. temperature difference $t_2 - t_1$.

H = heating surface of the apparatus in square meters.

Q = quantity of heat required in calories per minute.

$$Q = HC\,(t_2 - t_1) \qquad (2)$$

being the heat set free by condensation of the heating steam and passing through the walls into the boiling liquor.

Q_t = total heat. According to Regnault, to change one kilogram of water at 0° C. into saturated steam at the temperature t_1, there are required $606.5 + 0.305\ t_1$ calories.

Q_l = latent heat which is given off by condensation in the heating coils, according to Regnault, $= 606.5 - 0.695 t_2$ calories.

W_s = weight of the steam condensed in the heating tubes, in kilograms.

W_w = weight of the water evaporated by the heating surface.

Now let us consider copper coils of 75 mm. diameter and 20 meter lengths (in this case two coils where the inner coil has a smaller diameter than the outer coil), then according to formula (1)

$$K_v = \frac{1900}{\sqrt{dl}} = 1545$$

For practical use we take only two-thirds of this or 1030 and reducing it further in consideration of the dissolved substance by 20 per cent we get 824 calories per square meter per hour, which will give C = 13.73 calories per minute. The heating surface of the two coils, not taking the small thickness of the copper into consideration, will be approximately 10 square meters. We now have

$$Q = W_s Q_l \qquad (3)$$

$$W_s = Q \div Q_l \qquad (4)$$

$$Q = W_w\,(Q_t - t_0) \qquad (5)$$

$$W_w = Q \div (Q_t - t_0) \qquad (6)$$

Since $t_0 = 45°$, C., $t_1 = 60°$, C., $t_2 = 120°$, C., $t_2 - t_1 = 60°$ C., H = 10 sq. meters and C = 13.73 calories per minute, we find $Q = 10 \times 13.73 \times 60 = 8238$ calories per sq. meter and minute.

$$W_s = \frac{8238}{606.5 - (0.695 \times 120)} = 15.74 \text{ kgr.}$$

$$W_w = \frac{8238}{606.5 + (0.305 \times 60) - 45} = 14.21 \text{ kgr.}$$

We see from the above results that in a single effect having 10 square meters heating surface 14.21 kgr. of water can be evaporated per minute, if the liquor enters at 45° C. To

evaporate this quantity, 15.74 kgr. of steam are necessary, or 1 kgr. of heating steam evaporates 0.90 kgr. water per minute, as the heat of the condensed steam given off through the walls is the evaporating power for the liquor surrounding the heating tubes.

For the evaporation of acids, mostly single effects are used and the heating coils are lead or lead-lined copper coils.

When using lead heating tubes, the heat transmission coefficient is only about 45 per cent of the value for copper tubes, or C = 5.79 calories per minute. Then we get

$$Q = 10 \times 5.79 \times 60 = 3474 \text{ calories.}$$

$$W_s = \frac{3474}{606.5 - (0.695 \times 120)} = 6.64 \text{ kgr.}$$

$$W_w = \frac{3474}{606.5 + (0.305 \times 60) - 45} = 5.99 \text{ kgr.}$$

so that 10 square meter lead heating surface will evaporate under the same conditions only 5.99 kgr. water from the liquid per minute, and that to accomplish this 6.64 kgr. of heating steam are necessary.

Now let us suppose that a certain quantity of acid liquors has to be evaporated and the size and the heating surface of the single effect has to be calculated, then we can do this by the following method.

First of all, of course, the quantity of the liquor, the density of the thin liquor and the density of the concentrated liquor must be given.

Design of Tartaric Acid Evaporator.

In our case we will assume 1000 kgr. tartaric acid liquor which has a density of 9° Beaumé and has to be concentrated to 35° Beaumé. Tartaric acid liquor contains at 9.1° Beaumé 14 per cent solids and at 35° Beaumé 57.9 per cent solids. The specific heat, which we will need later on, is 0.911 at 9.1° Beaumé and at 18° C. according to Julius Thomsen, 18° C.= 0.911 and from his results we find the specific heat for 35° Beaumé to be 0.668.

The water to be evaporated can be found by the following formula:

$$W_w - W_c = W_w \left(1 - \frac{d_1}{d_2}\right) \qquad (7)$$

Where

W_w = the weight of the liquor to be concentrated = 1000 kg.

W_c = the weight of the concentrated liquor.

d_1 = percentage of solids in W_w = 14 per cent,

d_2 = percentage of solids in W_c = 57.9 per cent, therefore

$$W_w - W_c = 1000 \left(1 - \frac{14}{57.9}\right) = 760 \text{ kg. of water.}$$

We have to find now the quantity of calories needed to evaporate 760 kgr. of water from the following formula:

$$Q = W_w C_t + W_c S_1 t_1 - W_w S_0 t_0 \qquad (8)$$

Where

C_t = total heat, according to Regnault = 606.5 + 0.305t.

S_0 = specific heat of the thin liquor = 0.911.

S_1 = specific heat of the concentrated liquor = 0.668.

t_0 = temperature of the thin liquor in degrees C. = 45°.

t_1 = boiling temperature of the liquor in degrees C. = 60°

Inserting the numerical values in equation (8) we get Q = 614,266 calories.

To obtain the heating surface H for this single effect we can use the following formula, which follows directly from equation (2).

$$H = \frac{Q}{C\,(t_2 - t_1)} \qquad (9)$$

Where

Q = the heat passing through the heating surface in calories.

C = the heat transmission coefficient per square meter and hour = 1545 × 0.3 = 463.5 calories (1545 being the heat transmission coefficient for copper tubes and 0.3 the reduction factor for lead-copper tubes).

t_2 = temperature of the heating steam = 120° C.

t_1 = temperature of the boiling liquor = 60° C.

$$H = \frac{614{,}266}{463.5 \times (120 - 60)} = 22.08 \text{ square meters.}$$

If lead-lined copper tubes are used for heating coils, which are very advantageous, as they have a larger heat transmission coefficient than lead alone, we can take the heat transmission coefficient that of copper and lead combined.

Design of Quadruple Effect.

I will now take up the calculation of a multiple effect, say a quadruple effect. I will fix the size of the heating surface of the first body, and accordingly find the heating surfaces of the second third and fourth bodies. In this case I will first give a general solution and then reduce the heat transmission coefficient to say 25 per cent to take account of the dissolved substances in the liquor.

In designing an evaporator, the most important question is as to the design which will be able to evaporate the most water with the least quantity of steam.

The cheapest steam is always the exhaust steam from the engine or pumps, assuming, of course, that a steady supply and pressure can be relied upon.

The exhaust steam economically discharged from any engine has about 0.5 atmosphere pressure, corresponding to a temperature of 112° C. The liquor in the first body with this heating steam is boiling at 99° C. and gives off steam of the same temperature for heating the second body. The liquor boils there at 86° C. and gives off steam of the same temperature for the heating of the third body, where the liquor boils at 73° C. Vapors of the same temperature pass over into the fourth body and cause boiling at 60° C.

The temperature difference between the heating steam of 112° C. and the boiling point of the fourth body is 52° C. To get an equal temperature difference in the four bodies we divide the 52° C. by four and get 13° C. as the difference in the temperature for each body.

The vacuum in the four bodies according to their respective boiling points will be the following:

	Boiling point.	Vacuum.
First body	99° C.	116 mm
Second body	86° C.	322 mm
Third body	73° C.	484 mm
Fourth body	60° C.	611 mm

By selecting an evaporator for the first body containing one hundred 50-mm diameter copper tubes, 4 meters long each, we have a heating surface of 68 square meters. The second, third and fourth bodies will be found by calculation.

The heat transmission coefficient for the size of copper tube equals 4253 calories per square meter and hour for 1° C., but for liquors containing about 25 per cent dissolved substances only one-half of the above coefficient can be assumed. We further suppose that the liquor to be evaporated enters the first body at a temperature of 45° C. The calculations are then as follows:

First Body.

t_0 = 45° C. (temperature of entering liquor).

t_1 = 99° C. (boiling temperature).

t_2 = 112° C. (temperature of heating steam).

$t_2 - t_1 = 112 - 99 = 13°$ C. (temperature difference).

H_1 = 68 square meters (heating surface).

$C = \frac{4253}{2} = 2126.5$ calories (heat transmission coefficient).

$Q_1 = H_1 C\,(t_2 - t_1)$ = quantity of calories passing through the heating surface = 68 × 2126.5 × 13 = 1,879,826 calories.

$Q_{t_1} = 606.5 - 0.695t_2 - 0.00002t_2^2 - 0.000000,3t_2^3 = 528$ cal.

$Q_{t_1} = 606.5 + 0.305\,t_1 = 636.69.$

$Q_{t_1} - t_0 = 606.5 + (0.305 \times 99) - 45 = 591.69.$

$$W_{s1} = \frac{Q_1}{Q_{t_1}} = \frac{1,879,826}{528} = 3560.29 \text{ kgr. condensed steam.}$$

$$W_{w1} = \frac{Q_1}{Q_{t_1} - t_0} = \frac{1,879,826}{591.69} = 3177.04 \text{ kgr. evaporated water.}$$

Second Body.

$t_0 = 99°$ C.

$t_1 = 99 - 13 = 86°$ C.

$t_2 = 99°$ C.

$t_2 - t_1 = 13°$ C.

$Q_{t_2} = 606.5 - 0.695t_2 - 0.00002t^2_2 - 0.0000003t^3_2 = 537.2$ cal.

$Q_{t_2} - t_0 = 606.5 + (0.305 \times 86) - 99 = 533.73$ calories.

$Q_2 = W_{w1}\,Q_{t_2} = 3177.04 \times 537.2 = 1,706,705.88$ calories.

$$H_2 = \frac{1,706,705.88}{2126.5 \times 13} = 61.73 \text{ square meters.}$$

$W_{s2} = 3177.04$ kgr. condensed steam.

$$W_{w2} = \frac{1,706,705.88}{533.73} = 3197.69 \text{ kgr. evaporated water.}$$

Third Body.

$t_0 = 86°$ C.

$t_1 = 86 - 13 = 73°$ C.

$t_2 = 86°$ C.

$t_2 - t_1 = 13°$ C.

$Q_{t_3} = 606.5 - 0.695t_2 - 0.00002t^2_2 - 0.0000003t^3_2 = 546.39.$

$Q_{t_3} - t_0 = 606.5 + (0.305 \times 73) - 86 = 542.7$ calories.

$Q_3 = W_{w2}\,Q_{t_3} = 3197.69 \times 546.39 = 1,746,785.84$ calories.

$$H_3 = \frac{1,746,785.84}{2126.5 \times 13} = 63.18 \text{ square meters.}$$

$W_{s3} = 3197.69$ kgr. condensed steam.

$$W_{w3} = \frac{1,746,785.84}{542.76} = 3218.33 \text{ kgr. evaporated water.}$$

Fourth Body.

$t_0 = 73°$ C.

$t_1 = 60°$ C.

$t_2 = 73°$ C.

$t_2 - t_1 = 13°$ C.

$Q_{t_4} = 606.5 - 0.695t_2 - 0.00002t^2_2 - 0.0000003t^3_2 = 555.54$ cal.

$Q_{t_4} - t_0 = 606.5 + (0.305 \times 60) - 73 = 551.8$ calories.

$Q_4 = W_{w3}\,Q_{t_4} = 3218.33 \times 555.54 = 1,787,911$ calories.

$$H_4 = \frac{1,787,911}{2126.5 \times 13} = 64.67 \text{ square meters.}$$

$W_{s4} = 3218.33$ kgr. condensed steam.

$$W_{w4} = \frac{1,787,911}{551.8} = 3240 \text{ kgr. evaporated water.}$$

To summarize the results, we have found:

Body	Heat units in calories	Heating surface in square meters	Condensed steam in kgrs.	Evaporated water in kgrs.
1.....	1,879,826.00	68.00	3,560.27	3,177.04
2.....	1,706,705.88	61.73	3,177.04	3,197.69
3.....	1,746,785.84	63.18	3,197.69	3,218.33
4.....	1,787,911.00	64.67	3,218.33	3,240.00
	7,121,228.72	257.58	13,153.33	12,833.06

In this quadruple effect 12,833.06 kgr. water can be evaporated from the thin liquor per hour with a heating surface of 257.08 square meters, or 49.82 kgr. per square meter per hour.

To evaporate this quantity 3560.27 kgr. steam have to be utilized in the first body, consequently one kilogram of steam can evaporate 3.5 kilograms of water.

In the above outline, giving the general calculations, we did not take into consideration the heat radiated from the surfaces of the different effects which cannot be prevented entirely if even some covering is used.

The so-called self evaporation was also neglected, which goes on in the second, third, and fourth bodies. When the heating steam and liquor enter the second body from the first body, they have the same temperature, but, on account of the higher vacuum there, the liquor will naturally boil at a lower temperature. The liquor has the boiling temperature of the first body, consequently it drops here to the temperature of the lower boiling point, under a higher vacuum, and gives off the difference of the heat which helps the evaporation. Or, in other words, it evaporates itself to a certain degree and this we call *self-evaporation.*

As it is well known, the theoretical results always need some corrections, and so we would advise to correct the calculated heating surfaces for the different bodies, taking the circumstances of the case into proper consideration. By an increase of about 25 per cent or more in a similar case, for the different bodies we will get a result on which we surely can rely as satisfactory when designing the quadruple effect.

Design of Quadruple Effect for Recovery of Potash from Wool Waste Liquor.

We will now consider another case in which a certain quantity of liquor is given and all the data of a quadruple effect have to be calculated. It has to handle 50,000 kgr. wool waste liquor (from the wool washing process) for the purpose of the recovery of potash, which is well worth recovering.

This liquor contains the following constituents or solids dissolved:

Potassium carbonate (K_2CO_3)	1.63	per cent
Potassium chloride (K Cl)	0.18	per cent
Potassium sulphate (K_2SO_4)	0.236	per cent
Sodium carbonate ($NaCO_3$)	0.129	per cent
Grease	0.249	per cent
Organic impurities	0.696	per cent
Insoluble matters	0.15	per cent
	3.27	per cent

It has a density of 3.70° Baumé.

This liquor should be evaporated to 40.8° Baumé so as to contain:

Potassium carbonate	28.60	per cent
Potassium chloride	3.15	per cent
Potassium sulphate	4.13	per cent
Sodium carbonate	2.26	per cent
Grease	4.38	per cent
Organic impurities	12.22	per cent
Insolubie matters	2.64	per cent
	57.38	per cent

Hence from these 50,000 kgrs. of liquor we have to evaporate

$$W_w = W\left(1 - \frac{D_1}{D_4}\right)$$

$$= 50,000\left(1 - \frac{3.27}{57.38}\right) = 47,155 \text{ kgr. water}$$

where

W_w = weight of the evaporated water.

W = weight of the thin liquor.

D_1 = density of the thin liquor.

D_4 = density of the concentrated liquor.

So that the weight of the concentrated liquor becomes

$$W_c = W - W_w. \qquad (11)$$

$$= 50,000 - 47,155 = 2845 \text{ kgrs.}$$

Before going further in the calculation we have to decide on the pressure differences in the different bodies. Although some authorities state that the fall of the temperature between the heating chamber and juice spaces of the different

bodies must be constant, yet it will be shown that it must not be so in order to get the right results.

We will make the difference of pressure in the different bodies constant. We suppose that the liquor boils in the first body at 100° C. and in the fourth body at 60° C., then the total difference will be

$$1.00 - 0.196 = 0.804 \text{ atm.}$$

Therefore the pressure difference in each body will be: $0.804 \div 3 = 0.268$ atmosphere, and the pressure will be in the

First body	1.00	atmosphere
Second body	$1.00 - 0.268 = 0.732$	"
Third body	$0.732 - 0.268 = 0.464$	"
Fourth body	$0.464 - 0.268 = 0.196$	"

According to these pressures the boiling points and vacuums in the different bodies will be as follows:

	1 Body	2 Body	3 Body	4 Body
Pressure	1 atm.	0.732 atm.	0.464 atm.	0.196 atm.
Temperature	100° C.	90.27° C.	79.65° C.	60° C.
Vacuum	0	233.84 mm	407.12 mm	611.21 mm

Quantities of Water to Be Evaporated in the Different Bodies.

To evaporate the 47,155 kgrs. water from the 50,000 kgr. thin liquor so as to increase the density from 3.27 per cent to 57.38 per cent, according to Hausbrand the total quantity evaporated in the fourth body of a quadruple effect is 0.284 of the total evaporation

$$= 0.284\ (50{,}000 - 2845)$$
$$= 47{,}155 \times 0.284 = 13{,}392.02 \text{ kgr.}$$

With this data we will now figure the probable quantities evaporated in the other bodies, using Regnault's formulas:

C_t = total heat $= 606.5 + 0.305t$.

C_l = latent heat $= 606.5 - 0.695t - 0.00002t^2 - 0.0000003t^3$.

C_s = sensible heat $= t + 0.00002t^2 + 0.0000003t^3$.

Consequently

	1 Body.	2 Body.	3 Body.	4 Body.
Temperature	100° C.	90.27° C.	79.65° C.	60° C.
Total heat	637.0	634.03	630.79	624.8
Latent heat	536.5	543.38	550.87	564.7
Sensible heat	100.5	90.65	79.92	60.13

Having determined the basis for the quantity of water to be evaporated in the fourth body, we can start to calculate the third body.

It is plain that the sum of the heat in the concentrated liquor leaving the fourth body, and the vapor passing into the condenser and of the condensed water leaving the heating chamber, must be equivalent to the heat of the liquor leaving the third body and the heat of the evaporated water entering as vapor the heating chamber of the fourth body.

Following Francis Maxwell we can write now the following formula:

$$(W_{w4} + W_o)\, S_{h3} t_3 + W_{w2} C t_3 = W_o S_{h4} t_4 + W_{w4} C t_4 + W_{w3} C_{s3}$$

where

W_w = weight of the evaporated water in each individual body.

W_o = weight of the concentrated liquor.

S_h = specific heat of the liquor.

t = the boiling point in the bodies in degrees C.

C_t = total heat of vapor in calories.

C_s = sensible heat of condensed steam in calories.

C_l = latent heat of vapor in calories.

From our former calculations we see that the total heat of the evaporated water of the third body, forming the heating vapor of the fourth body ($W_{w2} C t_3$) less the sensible heat of this, condensed to water in the heating chamber of the fourth body ($W_{w3} C_{s3}$) must be the latent heat of the heating vapor in the fourth body ($W_{w3} C l_3$). Hence, from the above formula

$$W_{w3} C l_3 = W_{w4} C t_4 + W_o S_{h4} t_4 - (W_{w4} + W_o)\, S_{h3} t_3$$

$$W_{w3} = \frac{W_{w4} C t_4 + W_o S_{h4} t_4 - (W_{w4} + W_o)\, S_{h3} t_3}{C l_3} \qquad (12)$$

S_{h4}, S_{h3} and W_{w2} must now be found, where S_h = the specific heat. The specific heat of any solution can be found from the sum of the specific heats of the water and the dry substance it contains:

$$S_h = \frac{(100 - D)\, C_s}{100\, t} + \frac{S_d D}{100} \qquad (13)$$

D = percentage of dry substance.

C_s = sensible heat of water in calories.

S_d = specific heat of dry substance.

t = temperature of boiling in degrees C.

In our case the specific heats of the dry potash compounds in the wool waste-liquors are the following:*

1 Body.	2 Body.	3 Body.	4 Body.
100° C.	90.27° C.	79.65° C.	60° C.
at			
0.28237	0.28106	0.2797	0.2772

Using the above formula to determine the specific heat of the liquor in the fourth body, we get

$$S_{h4} = \frac{(100 - D_4)\, C_{s4} + S_{d4} D_4 t_4}{100\, t_4} \qquad (14)$$
$$= 0.5862 \text{ calorie.}$$

To figure S_{h3} it is necessary to obtain the density of the liquor leaving the third body, which is, according to the formula,

$$D_3 = \frac{W D_1}{W_{w4} + W_o} \qquad (15)$$
$$= \frac{163{,}500}{16{,}237} = 10.07 \text{ per cent} = 11.57° \text{ Baumé.}$$

From this we get the specific heat like before

$$S_{h3} = \frac{(100 - D_3)\, C_{s3} + S_{d3} D_3 t_3}{100\, t_3} \qquad (16)$$

in which $S_{d3} = 0.2797$, hence $S_{h3} = 0.9305$ calorie.

Now we can calculate the value of W_{w3}, inserting the numerical values found in equation (12)

$$W_{w3} = 13{,}186 \text{ kgr.}$$

In passing over to the evaporation in the second body, we have first to determine the density of the liquor leaving this body:

$$D_2 = \frac{W D_1}{W_{w4} + W_{w3} + W_o} \qquad (17)$$
$$= \frac{163{,}500}{29{,}423} = 5.55 \text{ per cent} = 6.36° \text{ Bé.}$$

The specific heat now is

$$S_{h2} = \frac{(100 - D_2)\, C_{s2} + S_{d2} D_2 t_2}{100\, t_2} \qquad (18)$$
$$= 0.96407 \text{ calorie.}$$

The evaporated water in the second body can be calculated by using the methods and different formulas of our first example for a quadruple effect. Thus the sum of the heat contained by the liquor entering the third body ($W_{w4} + W_{w3} + W_o$) $S_{h2} t_2$ and the heat of the vapors passing into the third body ($W_{w2} C t_2$) must equal the sum of the heat of the liquor leaving the third body ($W_{w4} + W_o$) $S_{h3} t_3$ and the heat and vapor leaving the third body ($W_{w3} t_3$), and the heat of the condensed heating steam ($W_{w2} C_{s2}$) leaving the third body.

$$(W_{w4} + W_{w3} + W_o)\, S_{h2} t_2 + W_{w2} C t_2 = (W_{w4} + W_o)\, S_{h3} t_3 + W_{w3} t_3 + W_{w2} C_{s2}$$

$$W_{w2} C t_2 - W_{w2} C_{s2} = (W_{w4} + W_o)\, S_{h3} t_3 + W_{w3} C t_3 - (W_{w4} + W_{w3} + W_o)\, C_{h2} t_2$$

$$W_{w2} C t_2 - W_{w2} C_{s2} = W_{w2} (C t_2 - C_{s2}) = W_{w2} C l_2$$

$$W_{w2} C l_2 = (W_{w4} + W_o)\, S_{h3} t_3 - (W_{w4} + W_{w3} + W_o)\, C_{h2} t_2$$

*For all the analytical and thermo-chemical data the author is indebted to Dr. Walter Glaeser, chemist of Chas. Pfizer & Co., Chemical Works, Brooklyn.

$$W_{w3} = \frac{(W_{w4} + W_c)\, S_{h3} t_3 + W_{w2} C_{t2} - (W_{w4} + W_{w2} + W_c)\, C_{h2} t_2}{C_{t2}} \quad (19)$$

Hence by substituting the numericals values, we get

$$W_{w3} = 12{,}809 \text{ kgr.}$$

Now we can pass over to the first body, but have first to determine the density of the liquor leaving it:

$$D_1^1 = \frac{W\, D_1}{W_{w4} + W_{w3} + W_{w2} + W_c} \quad (20)$$

$$= 3.87 \text{ per cent} = 4.5^\circ \text{ Baumé.}$$

$$S_{h1} = \frac{(100 - D_1^1)\, C_{s1} + S_{d1}\, D_1^1\, t_1}{100\, t_1} \quad (21)$$

$$= 0.977 \text{ calorie.}$$

We can use for the quantity of evaporation the same formula which was used for the second body:

$$W_{w1} = \frac{W_{w3} C_{t3} + (W_{w4} + W_{w3} + W_c)\, S_{h3} t_3 - (W_{w4} + W_{w3} + W_{w2} + W_c)\, S_{h1} t_1}{C_{t1}} \quad (22)$$

$$= 12{,}406 \text{ kgr.}$$

Summarizing the above results for the different bodies, we have:

1 Body	12,406 kgr.
2 Body	12,809 kgr.
3 Body	13,186 kgr.
4 Body	13,912 kgr.
	52,313 kgr.

which would mean the evaporation of more water from our liquor than it contains. However, we can make a correction, reducing the quantities for the four bodies in the same proportion, which is practically permissible. In this case by deducting 9.85 per cent the above results are reduced as follows:

1 Body 12,406 — 9.85 per cent = 11,184 kgr. water evaporated.
2 Body 12,809 — 9.85 per cent = 11,547 kgr. water evaporated.
3 Body 13,186 — 9.85 per cent = 11,887 kgr. water evaporated.
4 Body 13,912 — 9.85 per cent = 12,541 kgr. water evaporated.

47,159 kgr.

which nearly equals the total weight of the water to be evaporated. From these results we can figure the heat consumption for each body.

But before doing so, it is important to consider the loss of heat from the bodies and their pipe connections by radiation and by the contact with the surrounding air.

Radiation and Connection Heat Losses.

For our example we are going to use wrought-iron evaporating bodies 5 meters long and 3 meters diameter each. The exposed surfaces we will figure as 65 square meters for each body.

According to Peclet those losses are expressed in the following formula:

$$M = R + A \quad (23)$$

where M is the total loss of heat per hour and per square meter in calories, R the loss by radiation and A the loss to the surrounding air. Peclet's empirical formulas are:

$R = 124.72\, K\, a\, \Theta\, (a^t - 1)$, where

Θ = temperature of the air,

t = difference in temperature between the hot body and the air,

$a = 1.0077$, and for wrought iron,

$K = 2.77$;

further $A = 0.552\, K_1\, t^{1.233}$ and in our case

$K_1 = 2.13$.

Substituting the numerical values, we get

$$M = R + A = 124.72\, K\, a\, \Theta\, (a^t - 1) + 0.552\, K_1\, t^{1.233} \quad (24)$$

According to this formula we find the following results for our case in Peclet's table:

For the first body with 70° C. temperature difference between the body and the surrounding air:

$M_1 = 221 + 213 = 434$ calories per square meter.

The exposed surface being 65 square meters, the total loss L_4 is

$L_1 = 65 \times 434 = 28{,}210$ calories.

$M_2 = 183 + 176.3 = 359.3$ calories per square meter.

$L_2 = 359.3 \times 65 = 23{,}354.5$ calories.

$M_3 = 146 + 140.5 = 286.5$ calories per square meter.

$L_3 = 286.5 \times 65 = 18{,}622.5$ calories.

$M_4 = 76.7 + 75 = 151.7$ calories per square meter.

$L_4 = 151.7 \times 65 = 9860.5$ calories.

Heat Consumption of Different Bodies.

We can now proceed to find the total heat consumption for each body by calculating the utilized heat. The total amount of heat of a certain liquor in a certain body is equal to the heat contained by this liquor and the heat which is utilized to reach a certain evaporation.

For the first body of this quadruple effect this will be as follows:

$$W_{w1}\, C t_1 + (W_{w4} + W_{w3} + W_{w2} + W_c)\, S_{h1}\, t_1 = W_w\, S_{h0}\, t_0 + Q_{u1}$$

where

Q_{u1} = utilized heat in first body in calories.

S_{h0} = specific heat of the entering liquor in the first body.

t_0 = temperature = 80° C. entering liquor in the first body.

$$Q_{u1} = W_{w1}\, C t_1 + (W_{w4} + W_{w3} + W_{w2} + W_c)\, S_{h1}\, t_1 - W_w\, S_{h0}\, t_0 \quad (25)$$

$$S_{h0} = \frac{(100 - D_0)\, C_{s0} + S_{d0}\, D_0\, t_0}{100\, t_0} \quad (26)$$

$$= 0.979 \text{ calorie.}$$

By inserting the numerical values in (25) we now get

$$Q_{u1} = 7{,}000{,}922 \text{ calories.}$$

and the total heat consumption for the first body: Q^1

$$Q^1 = Q_{u1} + L_1 \quad (27)$$

$$= 7{,}000{,}922 + 28{,}210$$

$$= 7{,}029{,}132 \text{ calories per hour.}$$

The total heat consumption for the next bodies can be calculated similarly as we did it in our first example of the quadruple effect, namely by multiplying the vapor of the preceding body with the corresponding latent heat, and adding to it the corresponding losses by radiation as found before.

$$Q_{u2} = 11{,}184 \times 536.5 = 6{,}000{,}216$$

$$Q^{II} = Q_{u2} + L_4 \quad (28)$$

$$= 6{,}023{,}570.5 \text{ calories per hour.}$$

$$Q_{u3} = 11{,}547 \times 543.38 = 6{,}274{,}408.86.$$

$$Q^{III} = Q_{u3} + L_3 \quad (29)$$

$$= 6{,}293{,}031.36 \text{ calories per hour.}$$

$$Q_{u4} = 11{,}887 \times 550.87 = 6{,}548{,}191.69.$$

$$Q^{IV} = Q_{u4} + L_4 \quad (30)$$

$$= 6{,}558{,}052.19 \text{ calories per hour.}$$

The total heat consumption of the different bodies is, therefore, as follows:

First body	Q^I	= 7,029,132 calories
Second body	Q^{II}	= 6,023,571 calories
Third body	Q^{III}	= 6,293,031 calories
Fourth body	Q^{IV}	= 6,558,052 calories

Self-Evaporation.

Self-evaporation has already been mentioned, and in this case we will consider it and the resulting heat-unts will have to be deducted from the total heat consumption of the second, third, and fourth bodies.

Self-evaporation for the second body:

$$W_{s2} = (W - W_{w1})\, (t_1 - t_2)\, S_{h1} \quad (31)$$

Where

$W - W_{w1}$ = the liquor leaving the first body,

$t_1 - t_2$ = temperature difference between the first and second bodies.

$$W_{w_2} = (50{,}000 - 11{,}184)\ (100 - 90.27) \times 0.977$$
$$= 368{,}752 \text{ calories.}$$
$$W_{w_3} = (W - W_{w_1} - W_{w_2})\ (t_2 - t_3)\ S_{h_2} \qquad (32)$$
$$= 278{,}962 \text{ calories.}$$
$$W_{w_4} = (W - W_{w_1} - W_{w_2} - W_{w_3})\ (t_3 - t_4)\ S_{h_3} \qquad (33)$$
$$= 281{,}183 \text{ calories.}$$

If these values are now deducted from the total heat consumptions of the different bodies we get the corrected quantities of heat units passing through the heating surfaces.

First body ... $Q_1 =$ 7,029,132 calories.
Second body . $Q_2 = 6{,}023{,}570.5 - 368{,}752 = 5{,}654{,}818.5$ calories.
Third body .. $Q_3 = 6{,}293{,}031.36 - 278{,}962 = 6{,}014{,}069.36$ calories.
Fourth body . $Q_4 = 6{,}558{,}052.19 - 281{,}183 = 6{,}276{,}869.19$ calories.

Calculation of Heating Surfaces.

The heating surface of an evaporating apparatus can be calculated, as we saw before, from the equation (9)

$$H = \frac{Q}{C\,(T - t)} \text{ where}$$

Q = quantity of heat passing through the heating surface in calories.

C = heat transmission coefficient per square meter and hour.

T = temperature of heating steam.

t = temperature of boiling liquor.

With regard to the heat transmission coefficient for a quadruple effect, it is different in each body on account of the difference in pressure. Further, because of its low pressure, the heating steam never flows constantly with the same velocity and sometimes even passes too slowly and has to be helped along by means of air-vents to make it move a little more rapidly. The cleaning condition of the heating steam pipes has also a great influence.

Dr. H. Claassen made experiments on sugar evaporation, and by using his and other observations Hausbrand has calculated the following ratios for a quadruple effect:

1 : 0.91 : 0.75 : 0.55

In our case the heat transmission coefficients, therefore, become:

First body ..... $C_1 = 2184$
Second body ..... $C_2 = 1987$
Third body ..... $C_3 = 1638$
Fourth body ..... $C_4 = 1200$

Assuming that the heating steam enters the heating surface of the first body at 1.5 atmospheres absolute pressure corresponding to a temperature of 112° C., then:

$$H_1 = \frac{7{,}029{,}132}{2184 \times (112 - 100)}$$
$$= 268 \text{ square meters.}$$
$$H_2 = \frac{5{,}654{,}818}{1987 \times (100 - 90.27)}$$
$$= 294 \text{ square meters.}$$
$$H_3 = \frac{6{,}014{,}069}{1638\ (90.27 - 79.65)}$$
$$= 346 \text{ square meters.}$$
$$H_4 = \frac{6{,}276{,}869}{1200 \times (79.65 - 60)}$$
$$= 266 \text{ square meters.}$$

On account of the above mentioned uncertain conditions of the heating surfaces, it is advisable to increase somewhat the sizes calculated from the formula. We make the following corrections:

$H_1 = 280$ square meters.
$H_2 = 310$ square meters.
$H_3 = 365$ square meters.
$H_4 = 280$ square meters.

Calculation of Connecting Pipes.

The most important parts of a quadruple effect are the vapor-connecting pipes between the different bodies. These can be easily calculated, as we have determined the evaporated quantities of water. For this purpose the following formula can be used:

$$Q = K\,A\,v \qquad (34)$$

Where

Q = the volume of steam in liters passing through the connecting pipes.

$$Q = \frac{W_w}{3600} \times \frac{V}{1000}$$

Where

W_w = weight of evaporated water in the bodies in kgr. per hour.

V = the volume of one kgr. vapor in liters.

K = coefficient, with an approximate value of 0.8.

A = cross-section of steam pipe

$$= \frac{d^2\,\pi}{4}$$

d being the diameter of the pipe.

v = velocity of vapor in meters per second.

$$\frac{W_w}{3600} \times \frac{V}{1000} = \frac{d^2\,\pi\,K\,v}{4} \qquad (35)$$

According to the experiments made by Regnault the approximate volume of one kgr. of vapor occupied in the different bodies at the existing pressures is:

First body ..... 1699 liters.
Second body ..... 2268 liters.
Third body ..... 3446 liters.
Fourth body ..... 7747 liters.

The velocity v of steam may be taken 30 meters per second for the first three connections and for the fourth discharge of vapors into the condenser as 50 meters per second.

From the above equation we get

$$d = 0.121 \sqrt{W_w\,V} \qquad (36)$$

for the first, second and third bodies, and

$$d = 0.094 \sqrt{W_w\,V} \qquad (37)$$

for the fourth body to the condenser; consequently

$$d_1 = 0.121 \sqrt{W_{w_1} V_1} = 528 \text{ mm.}$$
$$d_2 = 0.121 \sqrt{W_{w_2} V_2} = 619 \text{ mm.}$$
$$d_3 = 0.121 \sqrt{W_{w_3} V_3} = 774 \text{ mm.}$$
$$d_4 = 0.094 \sqrt{W_{w_4} V_4} = 926 \text{ mm.}$$

The practical diameters may be taken as follows.

$d_1 = 550$ mm.
$d_2 = 640$ mm.
$d_3 = 800$ mm.
$d_4 = 950$ mm.

Calculation of Furnace Charges.

By Regis Chauvenet.

Chemical Introduction.

The computation of slags, *given*: (1) the ore and fluxes by analysis, and (2) the composition of the slag sought, has, I believe, never been made the subject of a special treatise. Several works on the metallurgy of certain metals have devoted more or less space to the topic. As a rule, the methods have lacked generality, often they were hardly "methods" properly speaking, but rather illustrations of special cases, sometimes of the "cut and try" order. A singular mystery has been thrown about the subject, which in reality presents no especial difficulties.

The slag "sought" can be attempted, of course, only when composition of ore and fluxes is known. In the iron blast

furnace we have the simplest case, because of the absence of matte-forming material, and of any reducible metals other than iron. Hence it is far more practicable in that special case to produce, by a properly calculated charge, almost the exact slag desired.

Per contra, owing to the high temperature of the iron furnace, it is less imperative to produce a slag of low-fusion temperature, so that a greater latitude of composition is allowable than in the smelting of other metals. Analyses have been published showing astonishing variations in the silica percentages of iron slags, varying from the high "twenties" up to as much as 60 per cent in extreme instances. Those who can remember the old charcoal practice of the Southern states, now nearly obsolete, may also call to mind the "slag wheel" by means of which the almost incredibly siliceous slag was dragged from the furnace in ropy and reluctant masses.

Some knowledge of chemistry is necessary in slag computation, and some familiarity with elementary mathematics. Efforts have indeed been made to dispense with both. The writer does not discuss either the advisability or the results of such attempts. The chemistry is limited to first principles, chiefly to the fundamental laws and their arithmetical interpretation. The mathematics are confined to the "Rule of three," i. e., ratio and proportion, and to the algebraic knowledge required to solve simultaneous equations of the first degree.

It is not without hesitation that the elements of chemical calculation are here introduced, as there are many to whom these will appear to be of too elementary a nature. However, there are many persons who have a fair knowledge of the elements of chemistry who are at a loss how to apply their knowledge when it comes to calculations of even a very simple kind. For such, a chemical epitome is prefixed, with apologies to such as do not need it.

It is convenient in the classification of slags (i. e., formulistically) to return to the old "dualistic" form. This has now become so far obsolete as to be almost unknown to the younger generation of students, and familiar only as a part of chemical history to older men.

The "dualistic" system originated with Berzelius. The transition period was from about 1850 to 1875. A few examples of the old and new formulas will serve to show the essential differences. To simplify the illustration we omit the changes in the atomic weights of the dyed elements. (Calcium 20, iron 28, etc.)

Compounds.	Old style.	New style.
Sulphuric acid	H_2O, SO_3	H_2SO_4
Tri-calcic phosphate	$3CaO$, P_2O_5	$Ca_3(PO_4)_2$
Boric acid	B_2O_3, $3H_2O$	(2) $B(OH)_3$
Ammonium nitrate	$(NH_4)_2O$, N_2O_5	(2) NH_4NO_3
Iron singulo silicate	$2FeO$, SiO_2	Fe_2SiO_4

The old system is certainly more cumbersome, and the theories upon which it rested belong to an early period of chemical development. But in a certain sense it may be well to point out that the old system is still in use, viz.: in the statement of many chemical analyses. Thus, if we have to state a phosphate analysis, with a view of presenting its percentage of available phosphoric acid, supposing tri-calcic phosphate to be one of the constituents, we would be forced to separate the "basic" and "acid" components, or the "oxide" and "anhydride" stating them as percentages, i. e., as CaO and P_2O_5, in spite of the fact that we no longer so write them in our accepted formulas.

Coming to the "silicates," which form about the only constituents of a slag, the old way of regarding the relation as one between the oxygen of the silica and the oxygen of the base still holds and is both convenient and practical. As based upon the consideration defined above, this nomenclature has the merit of great simplicity. Compare the oxygen in the silica with the oxygen in the base in the following illustrations, and no further explanations will be needed. We take the iron silicates as examples.

Formulas.	Names.
$(FeO)_2$, SiO_2	Singulo silicate.
FeO, SiO_2	Bi-silicate.
$(FeO)_3$ $(SiO_2)_2$	Tri-silicate.
$(FeO)_4$ $(SiO_2)_3$	Sesqui-silicate.

A glance at these formulas shows that when the oxygen of the silica equals that of the base we have the "singulo," when twice that of the base, the "bi-silicate" and so on, the three-to-two relation taking the usual designation of "sesqui."

Written in modern form these silicates would read as follows:

Fe_2SiO_4 ..... Singulo
$FeSiO_3$ ..... "Bi-"
$Fe_2Si_2O_6$ ..... "Tri-"
$Fe_4Si_3O_{10}$ ..... "Sesqui-"

Evidently if we are to see the "oxygen ratio" in complex formulas of the modern type, we must first "dissect" them.

So much for the ordinary slag nomenclature, in which we leave out for the present any discussion of the almost universal fact that in practice all slags are "polybasic."

To assume that the reader of this article is unacquainted with the elements of theoretical chemistry would be going too far. It is hardly possible to introduce the topic of calculations based on formulas and equations without making certain assumptions as to the fundamental knowledge of the reader, and as it is necessary to be definite in this respect, we here indicate what is supposed to be known, putting it into the form of condensed statements of laws and definitions, to refresh the memories of such as may not have had recent occasion to use them.

The definitions are conventional and wholly disregard all theories as to the nature of "atoms" and "molecules." In short, "stoichiometry," the "arithmetic of chemistry," is independent of the theory of the subject, but it is convenient, in fact almost inevitable, that it should make use of its language.

"Atomic weights" then, for our purpose, are the ratios in which, or in simple multiples of which, the elements unite among themselves.

The "molecular weight" of a substance is the sum of the atomic weights contained in its molecule.

The symbol of an element carries with it "understood" its atomic weight.

A chemical equation represents some chemical action. Its members must be equal in "weight" and in atoms of each element respectively ("balance"), thus illustrating the law that the mass of matter is not changed by a chemical reaction.

"Actual and atomic weights are proportional."

This simple dictum involves nearly all of the "mathematics" used in computing from chemical formulas. It need hardly be pointed out that the statement applies to "molecular weights" as well.

As it is unnecessary in slag work, to regard fractional values of the atomic weights, we shall use the following table in which only the nearest units are indicated.

Elements.	Atomic weights.	Elements.	Atomic weights.
Al. Aluminium	27	Pb. Lead	207
As. Arsenic	75	Mg. Magnesium	24
Ca. Calcium	40	Mn. Manganese	55
C. Carbon	12	O. Oxygen	16
Cu. Copper	64	Si. Silicon	28
H. Hydrogen	1	S. Sulphur	32
Fe. Iron	56	Zn. Zinc	65

The above are all of the elements we shall use in the examples to follow. The student should never attempt to "memorize" a table of this kind; this leads to confusion and error. By *using* it as a reference one soon associates the symbol and atomic weight properly.

It remains only to say that the subscript letters in formulas multiply the atomic weights of the elements to which they

are annexed. Thus Fe_2 goes into our calculation as 112; SiO_2 as 60; $CaCO_3$ as 100; $2CaO$ is the same as $(CaO)_2$; the comma will be used only to subdivide "dualistic" compounds, as for instance FeO, SiO_2 bisilicate of iron.

Chemical Introduction.

Chemical Arithmetic ("Stoichiometry") may be treated almost wholly by the use of the rules for "Ratio and Proportion." Knowledge of this and (later in the series) of simple equations, is assumed on the part of the reader.

Problem I. Given the weight of any substance forming one term of a chemical equation, to deduce the weight of any other term.

Examples. (a) $SiO_2 + 2FeO = (FeO)_2, SiO_2$.

If the SiO_2 (silica) weighs 50 lb., what will be the weight of the iron oxide which combines with it in this ratio (singulo silicate)?

Solution. Molecular weight of $SiO_2 = 28 + 32 = 60$.
Molecular weight of $2FeO = (56 + 16) \times 2 = 144$.

Since atomic (or molecular) weights represent actual numerical relations these molecular weights are in the same ratio as the *actual* weights in any given case, hence:

$$SiO_2 : 2FeO = SiO_2 : 2FeO.$$
$$60 : 144 = 50 = 120$$

It will then take 120 lb. of ferrous oxide to combine with silica in the singulo silicate ratio.

A very good, although quite conventional rule in working these proportions, is always to place the "theoretical" weights on the left and the actual weights on the right, bringing the unknown term last in the series, as in the above example. We use the equality sign rather than the "four dots" in our proportions. In fact, a proportion is nothing more than an expression of the equality between two ratios, and it might as well be so written.

Translated into ordinary language we say: "Molecular weight of silica is to molecular weight of two molecules of ferrous oxide, as actual weight of silica is to required weight of ferrous oxide."

(b) $Fe + H_2SO_4 = FeSO_4 + H_2$.

If we have 7 lb. of sulphuric acid, what weight of iron will it dissolve?

Atomic weight of iron = 56.

Molecular weight of sulphuric acid $= 2 + 32 + 64 = 98$.

It frequently happens that the ratios can be simplified. In the first example we might have written in the first member 5 : 12 instead of 60 : 144. In this case we write:

$$H_2SO_4 : Fe = H_2SO_4 : Fe$$
$$98 : 56 = 7 : 4 = 7 : 4$$

Answer—Four pounds of iron.

(c) One hundred pounds of lead sulphide is roasted to lead sulphate. What is the gain in weight?

$$PbS : PbSO_4 = 239 : 303$$
$$239 : 303 = 100 : 126.7$$

Answer—Weight of the sulphate 126.7 lb. Gain in weight 26.7 lb.

The following examples are given with answer, but no solution, as an exercise for the reader.

(d) An ore contains by analysis:

FeS_2	48.00 per cent
ZnS	24.25 per cent
SiO_2	27.75 per cent
	100.00 per cent

It is roasted, with reactions, as below:

$$4FeS_2 + 11\ O_2 = 2Fe_2O_3 + 8SO_2.$$
$$ZnS + 2\ O_2 = ZnSO_4.$$

Assume that 100 lb. was originally taken. What is the change in weight, after roasting?

Answer—No change in weight.

(e) $CaCO_3 = CaO + CO_2$.

This equation represents the burning of limestone to produce quicklime. We want 100 lb. quicklime (CaO). What weight of the limestone must we take?

$$CaO : CaCO_3 = 56 : 100$$
$$56 : 100 = 100 : 178+$$

Answer—178+ lb.

(f) Three tons of pyrite (FeS_2) are roasted to Fe_2O_3 (ferric oxide). What will be the weight of the latter?

Note that there is but one "Fe" in the formula of FeS_2 and two in the formula Fe_2O_3. We shall also find that we cannot equate this "roast" without taking eleven molecules of oxygen, thus:

$$4FeS_2 + 11O_2 = 2Fe_2O_3 + 8SO_2$$

Answer—Two tons.

Quite a variety of problems of metallurgical application can be brought under the operation of this general rule. It is only necessary to know the chemistry of the reaction, that is, by its equation. Sometimes it is not even necessary to understand the equation of reaction, see problem (c) above, in which we wrote simply the formulas for the original substance and that of the roasted product.

Problem II.—To find the percentage of any constituent in a chemical compound, the formula being given.

Example. (a) Find percentage of silica in the silicate (singulo) of iron $(FeO)_2, SiO_2$.

Solution.—First write out the theoretical composition in molecular weights, *e. g.*:

$$(FeO)_2 = 144$$
$$SiO_2 = 60$$

Molecular weight of compound = 204.

It is evident that the total "weight" is to the weight of silica as 100 is to the percentage of silica, that is:

$$204 : 60 = 100 : 29.4$$

Answer.—29.4 per cent silica.

This operation is too frequently performed and of too simple a nature to require the setting down of the proportion. *Divide molecular weight of the substance whose percentage is sought by the molecular weight of the whole compound*, then shift decimal point two places to the right, *i. e.*, multiply by 100.

By extending this to all of the constituents we get the "translation" of a formula into an analysis expressed in percentages.

(b) What is the percentage analysis of ferric oxide, Fe_2O_3?

Solution.—$Fe_2 = 56 \times 2 = 112$
$O_3 = 16 \times 3 = 48$

Molecular weight of ferric oxide = 160
Per cent of iron calculated as in (a) = 70.00
Per cent of oxygen calculated by same = 30.00

Summation 100.00

A few cases are appended, answers given, but no operations. The "problem" in fact hardly deserves the name.

(c) Find analysis of Dolomite; $CaCO_3 + MgCO_3$.

Divide the percentages as follows, viz., into CaO, MgO and CO_2.

Answer.	CaO	30.4 per cent
	MgO	21.7 per cent
	CO_2	47.8 per cent
	Total	99.9 per cent

(d) Find analysis of the mineral Cryolite; Na_3AlF_6.
(Atomic weights Na = 23. F = 19.)

Answer. $Na_3 = 32.8$ per cent. Al = 12.8 per cent. $F_6 =$ 54.3 per cent. Total 99.9 per cent.

(e) Find analysis of the silicate FeO, SiO_2.

Answer.—FeO = 54.5 per cent. $SiO_2 = 45.5$ per cent. Total 100 per cent.

Problem III.—To translate an analysis into a chemical formula. This may or may not be possible. If, however, the

substance analyzed is a true chemical compound, the transformation is not difficult.

Example.—(a) Take example (b) under the last problem and reverse it, that is, let composition by analysis be: Iron 70 per cent, oxygen 30 per cent. Divide each percentage by the atomic weight of the element represented, the quotients will obviously stand in the *relative* relations of the numbers of atoms present. Thus:

$$\frac{70}{56} = 1.250 \text{ and } \frac{30}{16} = 1.875.$$

If we take care to keep the decimal places the same in the two quotients we may drop the "points" as the ratio remains the same, so we write 1250 and 1875.

These numbers are evidently in the ratio of 2 to 3. We may write the formula: Fe_2O_3.

It is not always so easy to see the ratios, but by applying the rule of "greatest common diviser" any numbers which are in simple ratio can readily be reduced to their lowest terms.

Usually a slag analysis will not yield an exact formula, but it is often possible to find a close approximation.

Example.—(b) Analysis; $SiO_2 = 47$ per cent. $FeO = 38$ per cent. $CaO = 15$ per cent.

Apply the rule as above, we find as quotients:

$$(SiO_2)\ \frac{47}{60} = .783. \quad (FeO)\ \frac{38}{72} = .528. \quad (CaO)\ \frac{15}{56} = .268.$$

In cases like this, when we find that we can get no "greatest common divisor," a little judgment comes in. A person endowed with some "arithmetical perception" (a faculty hardly to be taught) will not be long in finding an adjustment which will lead to an approximate result. In the present instance, if we merely drop the third figures we find that we have to compare the numbers 78, 52 and 26, greatest common divisor = 26, so the three are in the simple ratio of 3:2:1. That is, we have three molecules of silica, two of iron oxide and one of lime, very nearly, and may write the formula:

$$2(FeO,SiO_2) + CaO,SiO_2.$$

The accurate analysis of the slag above was purposely thrown a little out of adjustment, to indicate how such approximations should be treated. No fixed rule can be given for such adjustments, if the numbers fail to yield an exact simple ratio. This case is really closer to a "formula" than should be expected in practice.

(c) Find formula for a compound analyzing:

$Al_2O_3 = 25.6$ per cent. $CaO = 14.1$ per cent. $SiO_2 = 60.3$ per cent.

Answer.—$Al_2O_3,(SiO_2)_3 + CaO,SiO_2$.

This problem admits of many interesting illustrations and extensions, but as its application in practical metallurgy is limited we content ourselves with these examples. The next one, however, is of constant use in a certain kind of slag computation and should be thoroughly understood before proceeding.

Its principle is the same as used in problem I.

Problem IV.—To calculate a chemical "excess" or "deficiency."

Example.—(a) The slagging constituents of a roasted ore are confined to silica and iron; analysis shows:

$SiO_2 = 40$ per cent, $Fe_2O_3 = 30$ per cent.

What is the silica excess in percentage over the formula $(FeO)_2SiO_2$?

Solution.—Molecular weight of $SiO_2 = 60$. Mol. wt. of $2FeO = 144$. Since the analysis shows ferric oxide, this must be reduced to ferrous oxide, their ratio being 160 to 144 or 10:9. This gives 27 per cent ferrous oxide.

We have then: $2FeO:SiO_2 = 2FeO:SiO_2$.

$$144:60\ (=12:5) = 27:11.25.$$

This shows the silica "required" for the ferrous oxide on the formula given. Actual silica is 40 per cent; subtracting the 11.25 we get 28.75 per cent.

Answer.—Silica excess = 28.75 per cent.

That is, in computing for a flux to be added we must provide for 28.75 lb. silica in each 100 lb. of the ore charged.

The next example is a "mixed" case, viz.: partly formulistic and partly arithmetical.

(b) An ore has as slagging constituents:

SiO_2, 30 per cent. FeO, 20 per cent. All other bases, 20 per cent.

We decide to flux on the convention that the FeO shall be to silica in the chemical ratio of $2FeO:SiO_2$, and that the weight of all the other bases shall be twice that of the silica provided for them. The latter part of the condition is what we term the "arithmetical" one. Find SiO_2 excess.

We start as before:

$$2FeO:SiO_2 = 2FeO:SiO_2$$
$$144:60\ (=12:5) = 20:8.33.$$

That is, the iron will take 8.33 per cent of the silica. Subtracting this from the 30 per cent of the analysis leaves 21.67 per cent silica, *available* for bases other than the iron. But as we require that all other bases shall be twice the weight of the silica provided for them, and as we have 20 per cent of these "other bases" in the ore, we subtract 10 per cent (half of twenty) more from the silica, so we have still left 11.67 *as "silica excess,"* and the amount of bases to be now provided for fluxing this ore will be 11.67×2 or 23.34 per cent. That is, using 100 lb. as unit weight of ore, 23.34 lb. base to be provided. *Answer.*—Silica excess = 11.67 per cent.

(c) Keep the statistics and results of example (b); let us further suppose that we have a limestone which contains:

Silica, 5 per cent. Lime (CaO), 53 per cent.

How much of this material will it take to satisfy the requirement of 100 lb. of the ore of example (b)?

Solution.—The 5 per cent silica of the limestone will call for twice its weight of lime, viz.: 10 per cent, leaving 43 per cent *available* lime for the silica of the ore. From last example we know that we need 23.34 lb. net base for each 100 lb. of ore, so the question is now reduced to this: "How many pounds of limestone with 43 per cent excess CaO, will furnish 23.34 lb. of the required (excess) base?"

$$43:100 = 23.34:54.$$

Answer.—54 lb. of limestone.

In any case similar to "c" a rule might be made, viz.: "Divide requirement by excess and multiply the result by 100."

(d) Ten pounds sulphuric acid dissolve 3 lb. of iron, how many pounds of zinc will the excess acid still dissolve?

Answer.—3.15 lb.

(e) An ore contains as "slag constituents" the following:

Silica	14.00 per cent
FeO	18.72 per cent
CaO	7.28 per cent

It is proposed to produce a singulo silicate slag. Adjudicated with this formula in view, what is in excess in the ore itself, by what weight, and of what substance. Assume 100 lb. of ore taken.

Answer.—Silica excess, 2.3 lb.

These elementary examples must serve as our chemical introduction. Their principles may be carried into far more complex problems.

If cases arise which the reader cannot state let him revert to the general rule, which is based on the fundamental facts of chemistry, viz.: "Atomic (or molecular) weights and actual weights are in the same ratios."

The method of application may not always be obvious, but in a great majority of cases this will be found to give the clue for a correct statement of conditions.

The calculations of furnace charges will be taken up in detail in several further articles.

Denver, Col.

Flow of Heat Through Contact Surfaces

By Carl Hering.

In the previous issue of this journal the writer published a table of values of the thermal resistivities of a number of materials, based on data gathered from various sources and there reduced to a common unit, the thermal ohm, that is, the degrees per watt.*

The purpose of the present article is to supplement this with a similar table, giving the values of the flow of heat through surfaces of contact between bodies and their equivalent thermal resistances, also collected from various sources and here reduced to watts and thermal ohms. Such figures are sometimes of importance in electric furnace work and in general in many other cases involving the flow of heat, when such a flow passes from one body to another through their surface of contact.

Such cases arise, for instance, in determining the heat lost to the air from the outside of a furnace wall, or from the top of a charge or a hot crucible, also the heat carried off by water cooling or in general when heating liquids, solids and gases in contact with one another.

A number of phenomena show that the thermal resistance of a contact between two surfaces may be very high, so high in fact that the resistances of the bodies themselves may sometimes be insignificantly small in comparison; it therefore at times becomes very important to take it into account in thermal problems. For instance, the contact resistance of the flames and the outside of the water tubes of an ordinary steam boiler, is extremely high; the heat gradient falls from the temperature of the hot flames to nearly that of the steam in a distance of apparently only a few thousandths of an inch, hence is an enormously steep heat gradient; a postage stamp pasted on the bottom of a tin cup in which water is boiled with a large Bunsen flame, will not even char, yet if increased to several thicknesses of stamps the outside ones will char, showing the extreme thinness of this high resistance layer. With such a high contact resistance no difference could be noticed between copper and iron pipes in boilers, although the latter have four times the thermal resistance of the former.

It is also known that a jointed graphite electrode may become red hot up to the joint and yet be black on the other side of it. It is said that invisible cracks in iron can be noticed by heating the piece so that the heat flows through it; it will flow readily up to a crack but only very slowly across it.

A recently described experiment (see this journal, November, 1911, p. 568) illustrated how very much greater the thermal resistance was from red-hot copper to the air, than through an equal section of copper.

This contact resistance from a hot body to the air may at times be sufficient to affect the temperature which the outside of a furnace will assume, hence, also, the amount of the loss through the walls. When the chief insulation resistance is in the walls this contact resistance on the outside will probably not be important, but when the flow of heat from the outside surface is great the contact resistance might become appreciable. Being a cheap form of resistance which occupies no space and weighs nothing, it might possibly even be made use of for insulation. One form in which it seems to be readily available is in the joints between the bricks or layers of the walls, instead of filling these solid with mortar, air spaces may be left to advantage and the bricks may preferably be laid with their flat sides perpendicular to the direction of flow so as to make as many joints as possible across the path. In one of the writer's furnaces constructed in this way the outside remained surprisingly cool even for high steel-melting temperatures inside.

Another instance of contact resistance and one which is of special importance in some electric furnaces is shown quite strikingly in the melting of scrap metals. When the pieces are forced into a bath of molten metal so as to become wetted by the liquid whereby the contact resistance is destroyed they take up the heat with far greater rapidity than when heated in the gases above the charge or by radiation from above; the difference is quite striking. Hence in a resistance furnace in which the heat is generated in the liquid metal it can be transmitted to the cold metal with very great rapidity and with practically perfect economy by thus eliminating the contact resistance, thereby reducing considerably the stand-by losses per ton and increasing the daily tonnage of the furnace.

Unfortunately our knowledge of such contact resistances and the laws governing them are very meager, hence our only recourse at present seems to be to take what experimental data exists and apply it as best we can. The table of data given below was compiled for this purpose and may be of value until better and more reliable experimental or analytical data become available. All the data have here being reduced to watts and thermal ohms, as these are the most rational units, being decimal multiples of the true absolute units. They are also the most convenient ones for the electric engineer for expressing a flow or transmission of energy in the form of heat; it is then in terms of the same unit in which the energy is expressed and measured in its electrical form, hence all the troublesome conversion factors are eliminated from the calculations. It is entirely unnecessary, besides being somewhat absurd, to change over from one unit to another and an incommensurate one at that, as the energy changes from the electric to the thermal form; it is the same energy, then why change units.

There are no less than 18 (perhaps even more) possible units in which a flow of heat can be expressed, beside 72 for thermal conductivity and an equal number for thermal resistivity, making a grand total of 162 possible units to choose from, and some writers seem to feel charitable toward those which they think have been slighted by others. Hence, by thus reducing all data to one standard uniform unit of heat flow, the watt, it becomes possible to compare the results of different observers directly, whereby a better choice of the more probably correct ones may be made.

Unfortunately it is quite common to use the word "radiation" very loosely and at times in a positively incorrect sense. In its correct sense it is used to distinguish the heat which is transmitted by true radiation from that transmitted by conduction and convection; notwithstanding this it is very often used for the total heat including that conducted and that conveyed. For instance, the losses through furnace walls governed probably almost entirely by conduction, are generally referred to as "radiation losses." A steam radiator is so called, although most of its heat is given off by convection. Some of the discrepancies in the data given in this table may, therefore, be due to some writers making the distinction and others being loose in their terms.

The term contact resistance has been used above as a convenient self-explanatory expression; whether the evidently great opposition which a flow of heat encounters at such surfaces, is a true resistance measurable in thermal ohms, may be open to question; the term virtual resistance would probably be a better one. It may, perhaps, be a different physical property, just as a counter e.m.f. in electric circuits is not a true resistance, although its value may, at times (but with caution), be expressed in its equivalent in ohms.

If this so-called thermal contact resistance were a true resistance, then the flow should increase in proportion with the temperature drop at the surface; that is, temperature drop/watts $= R$ should be constant. This is not the case, however, as the flow of heat from a surface to the air per degree of difference increases quite rapidly with the temperature; this is due in part to the fact that the radiation heat increases very rapidly with the temperature (with the fourth power of the absolute temperature and even faster), and whether it would be correct to speak of a resistance to radiant heat or not, is questionable, although its counterpart seems to be implied in radiant light in the terms translucent, trans-

*This journal, Dec., 1911, p. 652.

parent and opaque, and we know that a screen will intercept radiant heat by offering a resistance and it does not do this by radiating back an equal amount, but by an actual stoppage.

It is quite as correct, however, to express these surface resistances in thermal ohms as it is to reduce the heat flow to that per degree, which is done in many text books and tables. Hence the resistances have been added here for a square inch. They are merely the reciprocals of the watts per square inch, per degree, or the quotients of the temperature drop divided by the watts per square inch. They decrease very greatly when the temperatures become very high; for the theoretical "black body" this contact resistance may be zero in thermal ohms.

But, whether a surface resistance is a true resistance or not for practical purposes it is of service in simplifying calculations to express the resistivity (specific resistance) of a contact resistance in terms of a true thermal resistance like thermal ohms, and in order to correct it for higher temperatures it may be accompanied by a temperature coefficient, just as is done with the resistivities of solids. It would then have a negative coefficient, that is, it would decrease as the temperature rises, just as most refractory materials do; this decrease is shown very markedly in the table. But this is foreign to the present article.

To use such a figure for contact resistance one simply adds it to that of the body itself in order to get the total; then by dividing the total temperature drop (therefore including that at the surface) by this total resistance gives the total loss. In other words, it is treated simply as an ordinary resistance whose length (the thickness of the high resistance surface film) may be neglected. To find this contact resistance for a given surface, like the outside surface of a furnace, divide the thermal ohms per square inch as given in the table, by the total surface in square inches.

For a warm surface of ordinary iron, for instance, the surface resistance may be about 100 thermal ohms per square inch, hence for every square foot it would be $100/144 = 0.7$ thermal ohms, which is then added to that of the wall to get the total of that column of one sq. ft. section. The temperature drop will then, of course, be dependent on the flow of heat, and this should be of the order of about 0.4 watt per sq. in., hence about 58 per sq. ft. for which 100 thermal ohms was the contact resistance. It is, therefore, a calculation by trials, instead of a direct one, but this is true also of the flows through solid materials which also have a large temperature coefficient.

One of the complications which arises in reducing such thermal data as these to laws, is that when heat flows from a solid to air, as in exposed hot bodies, the flow *per degree* increases as the temperature rises, while when it flows in the reverse direction, from a gas to a solid, as in the case of the hot gases in a boiler heating the water tubes, just the reverse seems to be true. It has even been suggested to increase this heat flow per square foot through the tube walls by using lower temperature gases or liquids for heating the tubes.

Another complication in establishing such laws is shown in the spheroidal state. Ordinarily when heating water in an iron vessel, by means of very hot gases, as in a steam boiler, the high thermal resistance is on the fire side of the iron, but if given the opportunity it will quickly shift over to the water side, producing what is known as the spheroidal state, and it seems that it also increases as it changes its place.

Still another complication is due to the fact that when heat flows from a solid to a liquid, convection plays an important part, hence also the velocity of movement of the liquid. And with a gas there is radiation besides, which at times may become relatively large; in both there may or may not also be true conduction of the liquid or gas. Radiation per degree is known to increase rapidly as the temperature rises, but the laws concerning convection are not so definite, and in practical work we are generally concerned with the total and do not care for each one separately. Radiation laws apply strictly to a very useful theoretical creation called a "black body," namely one which absorbs all radiations falling on it and does not reflect or transmit any. For such bodies the radiated energy is proportional to the difference between the fourth powers of the absolute temperatures of the emitting and the receiving body. But for most actual bodies this must be multiplied by a very appreciable factor which is quite different for different materials and even varies very considerably with the temperature, all of which makes it necessary to go back to empirical data.

At best, therefore, any theoretical calculations of heat flows through contact surfaces would be likely to be quite complicated, and even then they might give very erroneous results owing to the apparently great differences between the physical conditions of the contact surfaces. For these and other reasons it will probably in most practical cases be best to use actual observed data, hence this table was prepared. It is also hoped that the inconsistencies in these results, which become more apparent by their direct comparison, as also the vagueness of some of the original data, will induce those who have the time and facilities to do so, to make more systematic and more complete re-determinations.

Owing to the shifting nature of this high resistance layer, generally to the gas side, the results are here arranged in various groups, as it may not always be proper to expect those in one group to agree with those in another. A number of data have been included for comparisons rather than for their direct use.

These tabulated results seem to enable the following general conclusions to be drawn; in doing this, however, the writer assumes no responsibilities for the correctness or reliability of the original data, but merely for their reduction to a uniform basis. References to all the original sources are given in the notes following the table.

Solid to solid. This applies to such cases as joints between bricks or layers, or between two parts of a jointed electrode, or between an electrode and its terminal, etc., in general, to any butted surfaces of solids which are not united by mortar, fluxing material, solder, etc., which destroy the contact resistance by making the parts mechanically continuous. No data could be found. It is known that for the good conductors like the metals, or graphite, the resistance of a crack is relatively very high. For insulators it seems also to be considerable, but as the material itself also has a high resistance the relative difference may be less. In water-cooling hot parts it seems important to avoid such abutting surfaces as much as possible between the wetted surface and the part to be cooled. Snyder* believes that the high resistance of joints between solids decreases very greatly at the higher temperatures.

Solid to gas. These results apply to such cases as the outside of furnaces, the cooling of the top surfaces of molten or hot charges, or of the outside of crucibles, etc. Only one contact surface is involved. They seem to show that with about ½ watt per sq. in., the surface is quite warm to the touch; with about 1 watt it would no doubt sizzle if wetted. With 3 or 4 watts, or roughly, half a kilowatt per sq. ft., it would be decidedly too hot to touch. At a dull red heat a metal surface exposed freely to the air would probably be losing about 7 to 11 kilowatts per sq. ft. At a bright red heat the loss from a metal surface or a hot crucible might be of the order of 12 to 15 kw per square foot; but with a still atmosphere of hydrogen, as in the molybdenum wire furnace, such very high temperatures as 1600° to 1700° C. are reached with only about 4½ to 8 kw per square foot, which is probably due more to the relative stillness of the heated gas than to its being hydrogen.

The results for the very thin wires from which the emission is enormous, due in part to their small diameters, are not

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directly comparable as the surface action of a very thin round wire seems to be different from that of a flat surface. It is of interest, however, that as high as 9000 kw per square foot are thus emitted to a gas, as distinguished from the usual vacuum. A vacuum reduces very greatly the flow from a surface, as is shown for instance by the rather low heat emission from the graphite surfaces in the Arsem furnace.

If the flow of heat through furnace walls amounts to 1 kw per square foot of inside surface (see this journal, November, 1911, p. 591), and if the flare to the outside is 3.5, then the flow of heat from the outside to the air would be $1/3.5 = 0.286$ kw per square foot. From the table it appears that this would be only about five times that escaping from dynamos when the rise of temperature is about 40°, less than half that for 200°, and only about double that from an ordinary steam radiator (see below); hence the outside of such a furnace would not be likely to get very hot, probably not much more than about 150° C. As this is a small drop of temperature compared with the total, this skin resistance is a small part of the whole, in this particular case.

When reduced to the heat emitted per degree, the results show that the earlier values of Peclet and of Box are either much too low or apply only to atmospheric temperatures; they may apply to true radiation only. They are of interest, however, in their relative values; they show how very greatly polishing a surface reduces its power to transmit heat to the air; it reduces it to about 1/6 to 1/7. This is also shown by the relative values for iron given at the end of this group; these also seem to show that oiling a polished surface increases the emission between two and three times, but has less effect on rough surfaces.

A fairly good, easily remembered approximate figure for ordinary iron surfaces for steam temperatures seems to be about 1/100 watt per C° per square inch. With from 3 to 15 watts per square foot per C°, the temperatures range from high steam heat to bright red heat for exposed surfaces, while with only 3 to 5 the temperature will be a white heat with a still gas, showing the very great effect of the movement of the gas. In a vacuum, about 4 watts per square foot per C° corresponds to 2000°.

Metal to Water.—These apply chiefly to water cooling, as for instance of electrode terminals. The data is very meager. It seems, however, that roughly as much as 100 kw per square foot of wetted surface can be relied upon without steaming, in brass cooling jackets, with but a moderate flow of water. Roughly this is about 1 kw per C° per square foot.

Water to Metal to Water.—Roughly about 1½ to 2 watts per square inch per C°.

Water to Metal to Air.—These apply more directly to hot water heating of rooms, and if it may be assumed that the high resistance is almost entirely on the air side, which is likely, then the results may be compared with the second group. About 50-90 watts per square foot seem to flow when the water is very hot. Reduced to 1°, a fairly good crude figure seems to be about 1 to 1½ watts per square foot; this increases with the temperature. Comparing these with those of the second group, they will be seen to be of about the same order of magnitude, which seems to show that the contact resistance resides almost entirely on the gas side.

Gas to Metal.—These apply to the heating of metals in flame furnaces, or to the flame side of steam boiler pipes. No data was found, but as has already been remarked there seems to be a very high contact or skin resistance, which in steam boilers is known to be remarkably high, and is very noticeable in the melting of metals, for when cold metal is submerged in liquid metal it acquires heat with far greater rapidity; theoretically most metals should sink in their melts, but in practice they unfortunately very often float on cushions of frozen high resistance slag, hence must be heated largely from the top by the gases and by radiation. It is believed by the writer that at first, when the metal is cold, this contact resistance when metals are melted with hot gases is approximately of the order of that in steam boilers discussed below, but probably somewhat less, and that it no doubt becomes much less per degree as the temperature of the solid metal rises.

Gas to Metal to Water.—These apply to steam boilers, condensers, etc. According to standard boiler practice the flow through the tubes is a little less than a kilowatt per square foot. Comparing this with the writer's rough figure for water cooling without steaming, namely 100 kw, it would seem that immensely more heat could be transmitted per square foot to the water in boilers without reaching the spheroidal state. Moreover 1 kw flowing through 1 sq. ft. of pipe, say ¼ in. thick, at about 0.8 thermal ohm, inch cube (see table in last month's issue), would mean only about 1.5° difference in temperature between the outside and inside, hence showing that the resistance is almost entirely on the outside between the gas and the metal, and that it is extremely high, indicating considerable room for improvement in steam boilers.

The single value given for this high temperature case per degree, for a locomotive boiler, roughly about 0.01 kw per square foot, seems to be in error, as it would, when compared with standard practice, mean only about 100° C. excess of temperature of the hot gases over that of the steam. But dividing the figure above it, which is standard practice, by say 1000° C. will give roughly 0.001 or only 1 watt per square foot per degree, which is only about 1/200 to 1/400 of the flow of heat through condenser tubes per degree, when the hot gas is steam. This again emphasizes the high resistance between a high temperature gas and relatively cold metal.

For condenser tubes the rate per degree is seen to be about 0.2 to 0.4 kw per square foot. The condenser tube rate is between 200 and 300 times as great as in hot water heating (see above), showing again the high resistance to air in the latter, as compared with water in the former.

Gas to Metal to Gas.—These refer chiefly to steam heating. Approximately the figures are about 100 to 200 watts per square foot, which is roughly from two to four times as great as for hot water heating; but this is due largely to the higher temperatures, for the values per degree are not very different. These latter values differ so greatly among themselves and depend so much on the movement of the air that a more careful comparison would hardly be warranted. But they seem to agree approximately with those for metal to air for similar temperatures, which would indicate that the high resistance was to the cold gas on the outside rather than from the hot gas to the metal on the inside, that is, the metal seems to tend to take the temperature of the hot gas rather than the colder one, when the latter is a gas, but the reverse when the colder body is water, as in steam boilers.

Reduction Factors.—The following reduction factors will enable those who prefer to use the older units to make the necessary reductions. 1 watt = 0.238882 small calories per second; 1 kw = 56.8776 b.t.u. per minute. 1 watt per square foot per C° = 1.90 b.t.u. per hour, per square foot, per F°. A drop in C° × 9/5 = the drop in F°.

FLOW OF HEAT THROUGH CONTACT SURFACES

Temperatures in Centigrade degrees

	Kw. per Sq. Ft.	Watts per Sq. In.	Therm. Ohms per Sq. In.	Watts per Sq. Cm.	Reference
Solid to solid, no data.					
Solid to gas (or vacuum):					
60° surfaces of dynamos to air at 20°	.058	.40	100.0	.062	Y
90° iron to air at 20°	.08	.56	126.0	.086	Sn
90° brick to air at 20°	.09	.63	112.0	.097	Sn
110° iron to air at 20°	.11	.76	118.0	.12	Sn
110° brick to air at 20°	.13	.90	100.0	.14	Sn
130° iron to air at 20°	.14	.97	113.0	.15	Sn
130° brick to air at 20°	.16	1.11	99.0	.17	Sn
150° iron to air at 20°	.18	1.25	104.0	.19	Sn
150° brick to air at 20°	.20	1.4	94.0	.22	Sn
170° iron to air at 20°	.22	1.5	98.0	.24	Sn

	Kw. per Sq. Ft.	Watts per Sq. In.	Therm. Ohms per Sq. In.	Watts per Sq. C m.	Reference
170° brick to air at 20°	.25	1.7	86.0	.27	Sn
190° iron to air at 20°	.27	1.9	91.0	.29	Sn
190° brick to air at 20°	.30	2.1	81.0	.32	Sn
220° iron to air at 20°	.34	2.4	85.0	.37	Sn
220° brick to air at 20°	.39	2.7	74.0	.42	Sn
200° nickel strip to air at 27°	.65	4.5	38.3	.70	X
300° nickel strip to air at 27°	1.39	9.7	28.3	1.5	X
1000° graphite to vacuum	2.19	15.2	65.8	2.36	A
400° nickel strip to air at 27°	2.32	16.1	23.2	2.5	X
500° nickel strip to air at 27°	3.90	27.1	17.5	4.2	X
1600° molybdenum ribbon to hydrogen	4.38	30.4	52.7	4.72	WD
600° nickel strip to air at 27°	6.31	43.8	13.1	6.8	X
2000° graphite to vacuum	7.63	53.0	37.8	8.21	A
1700° molybdenum wire to hydrogen	7.91	55.0	31.0	8.53	WD
700° nickel strip to air at 27°	9.86	68.4	9.8	10.6	X
800° nickel strip to air at 27°	12.2	84.5	9.1	13.1	X
2600° graphite to vacuum	13.1	91.0	28.6	14.1	A
900° nickel strip to air at 27°	13.4	92.9	9.3	14.4	X
Tungsten filament in lamp	51.8	360.0	....	55.8	M
Tantalum filament in lamp	54.6	379.0	....	58.8	M
927° tungsten wire 0.229 mm. diam. to hydrogen	66.0	458.0	2.0	71.0	L
Treated carbon filament in lamp	108.0	752.0	....	117.0	M
Metallized filament in lamp	117.0	812.0	....	126.0	M
Untreated carbon filament in lamp	136.0	947.0	....	147.0	M
727° tungsten wire 0.045 mm. diam. to hydrogen	138.0	960.0	.76	149.0	L
927° tungsten wire 0.069 mm. diam. to hydrogen	141.0	980.0	.94	152.0	L
927° tungsten wire 0.045 mm. diam. to hydrogen	198.0	1370.0	.68	213.0	L
1727° tungsten wire 0.229 mm. diam. to hydrogen	230.0	1590.0	1.1	247.0	L
Tungsten filament in nonoxidizing gas	337.0	2340.0	....	363.0	M
1727° tungsten wire 0.069 mm. diam. to hydrogen	458.0	3180.0	.54	493.0	L
1727° tungsten wire 0.045 mm. diam. to hydrogen	620.0	4300.0	.40	667.0	L
1727° tungsten wire 0.0088 mm. diam. to hydrogen	929.0	6540.0	.26	1000.0	L
2927° tungsten wire 0.028 mm. diam. to hydrogen	7120.0	49400.0	.06	7660.0	L
3127° tungsten wire 0.045 mm. diam. to hydrogen	9010.0	62500.0	.05	9700.0	L
Per degree:					
Silver, polished, to air	.0000140	.000097	10300.0	.0000150	P
Copper, polished, to air	.0000172	.000120	8340.0	.0000185	B
Tin, polished, to air	.0000232	.000161	6210.0	.0000250	B
Zinc and brass, polished, to air	.0000259	.000180	5560.0	.0000278	B
Tinned iron, polished, to air	.0000452	.000314	3180.0	.0000486	B
Sheet iron, polished, to air	.0000485	.000337	2970.0	.0000522	B
Sheet lead to air	.0000700	.000486	2060.0	.0000754	B
Sheet iron, ordinary, to air	.000298	.00207	483.0	.000320	B
Glass to air	.000313	.00218	460.0	.000337	B
Common steam pipe, inferred, to air	.000337	.00234	417.0	.000363	B
Cast iron, new, to air	.000341	.00237	423.0	.000367	B
Cast and sheet iron, rusted, to air	.000363	.00251	400.0	.000390	B
Sawdust, to air	.000379	.00263	380.0	.000408	P
Wood, building stone and brick, to air	.000389	.00270	370.0	.000418	B
Sand, fine, to air	.000389	.00270	370.0	.000418	P
Water to air	.000570	.00395	254.0	.000425	P
Oil to air	.000778	.00540	185.0	.000581	P
90° iron to air at 20°	.00114	.0079	126.0	.00123	Sn
110° iron to air at 20°	.00122	.0085	118.0	.00131	Sn
130° iron to air at 20°	.00127	.0089	113.0	.00137	Sn
90° brick to air at 20°	.00129	.0090	112.0	.00139	Sn
150° iron to air at 20°	.00139	.0096	104.0	.00149	Sn
188° iron to air at 20°	.00143	.0099	101.0	.00154	St
110° brick to air at 20°	.00144	.0100	100.0	.00155	Sn
60° surfaces of dynamos to air at 20°	.00144	.0100	100.0	.00155	Y
130° brick to air at 20°	.00145	.0101	99.0	.00157	Sn
170° iron to air at 20°	.00147	.0102	98.0	.00158	Sn
150° brick to air at 20°	.00154	.0107	94.0	.00166	Sn
190° iron to air at 20°	.00159	.0110	91.0	.00171	Sn
170° brick to air at 20°	.00167	.0116	86.0	.00179	Sn
220° iron to air at 20°	.00170	.0118	85.0	.00183	Sn
190° brick to air at 20°	.00177	.0123	81.0	.00190	Sn
220° brick to air at 20°	.00195	.0135	74.0	.00210	Sn
Enameled iron to air	.00216	.0150	66.7	.00232	Z
1000° graphite to vacuum	.00219	.0152	65.8	.00235	A
1600° molybdenum ribbon to hydrogen	.00274	.0190	52.7	.00295	WD
200° nickel strip to air at 27°	.00376	.0261	38.3	.00405	X
2000° graphite to vacuum	.00382	.0265	37.8	.00411	A
1700° molybdenum wire to hydrogen	.00466	.0323	31.0	.00501	WD
2600° graphite to vacuum	.00504	.0350	28.6	.00542	A
300° nickel strip to air at 27°	.00510	.0354	28.3	.00548	X
400° nickel strip to air at 27°	.00621	.0431	23.2	.00667	X
500° nickel strip to air at 27°	.00824	.0572	17.5	.00886	X
600° nickel strip to air at 27°	.01100	.0764	13.1	.0118	X
700° nickel strip to air at 27°	.01470	.102	9.8	.0158	X
800° nickel strip to air at 27°	.01580	.110	9.1	.0170	X
900° nickel strip to air at 27°	.01540	.107	9.3	.0166	X
927° tungsten wire 0.229 mm. diam. to hydrogen	.0712	.495	2.0	.0767	L
1727° tungsten wire 0.229 mm. diam. to hydrogen	.133	.921	1.1	.143	L
927° tungsten wire 0.069 mm. diam. to hydrogen	.152	1.06	.94	.164	L
727° tungsten wire 0.045 mm. diam. to hydrogen	.190	1.32	.76	.205	L
927° tungsten wire 0.045 mm. diam. to hydrogen	.214	1.48	.68	.230	L

	Kw. per Sq. Ft.	Watts per Sq. In.	Therm. Ohms per Sq. In.	Watts per Sq. Cm.	Reference
1727° tungsten wire 0.069 mm. diam. to hydrogen	.278	1.84	.54	.286	L
1727° tungsten wire 0.045 mm. diam. to hydrogen	.359	2.49	.40	.386	L
1727° tungsten wire 0.0088 mm. diam. to hydrogen	.538	3.79	.26	.580	L
2927° tungsten wire 0.028 mm. diam. to hydrogen	2.43	16.9	.06	2.62	L
3127° tungsten wire 0.045 mm. diam. to hydrogen	2.88	20.0	.05	3.10	L
Relative:					
Iron, rough, dry or oiled	100	...	..	...	...	M
Iron, planed, oiled	60	...	..	...	...	M
Iron, planed, dry	32	...	..	...	...	M
Iron, drawfiled, oiled	49	...	..	...	...	M
Iron, drawfiled, dry	20	...	..	...	...	M
Iron, polished, oiled	45	...	..	...	...	M
Iron, polished, dry	18	...	..	...	...	M
Metal to water:						
143° brass to water at about 40°	96.6	670.0	.154	104.0	H
Per degree:						
143° brass to water at about 40°	.938	6.50	.154	1.01	H
Water to metal to water, per degree:						
Water to iron plate to water	.211	1.47	.68	0.227	K6
Water to iron plate to water	.316	2.19	.46	0.340	K6
Water to metal to air:						
27° water to cast iron to air at 21°	.00464	.0322	173.0	.0050	S
43° water to cast iron to air at 21°	.0229	.159	140.0	.0247	S
107° water to cast iron to air at 20°	.0440	.306	258.0	.0473	K2
60° water to cast iron to air at 21°	.0446	.309	125.0	.0480	S
93° water to cast iron to air at 21°	.0952	.661	109.0	.102	S
Per degree:						
107° water to cast iron to air at 20°	.000506	.00388	258.0	.000544	K2
27° water to cast iron to air at 21°	.000836	.00580	173.0	.000900	S
43° water to cast iron to air at 21°	.00103	.00716	140.0	.00111	S
Water to iron to gas	.00105	.00730	137.0	.00113	K7
60° water to cast iron to air at 21°	.00115	.00799	125.0	.00124	S
93° water to cast iron to air at 21°	.00132	.00916	109.0	.00142	S
Water to iron to gas	.00264	.0183	54.7	.00284	K7
Gas to metal no data.						
Gas to metal to water:						
Fire gases to iron to steaming water (boiler)	.846	5.87	...	.910	K1
Per degree:						
Gas to iron plate to water (boiler)	.0090	.062	16.1	.010	K8
Steam to cast iron to water at 100°C.	.166	1.15	.87	.179	I
Steam to wrought iron to water at 100°C.	.197	1.37	.73	.212	I
100° steam to iron (vertical) to water (condenser)	.264	1.83	.55	.284	K4
Steam to brass to water at 100°C.	.294	2.04	.49	.316	I
Steam to copper to water at 100°C.	.339	2.35	.43	.365	I
100° steam to iron (horizontal) to water (condenser)	.386	2.68	.37	.416	K4
Gas to metal to gas:						
107° steam to bronzed cast iron to air to 20°	.073	.51	171.0	.079	K2
100° steam to iron to air at 20°	.095	.660	121.0	.102	K3
104° steam to cast iron to air at 21°	.115	.799	104.0	.124	S
127° steam to cast iron to air at 21°	.155	1.08	99.0	.167	S
153° steam to iron to air at 20°, observed	.170	1.18	113.0	.183	K5
153° steam to iron to air at 20°, calculated	.172	1.19	112.0	.185	K5
149° steam to cast iron to air at 21°	.199	1.38	93.0	.215	S
Per degree:						
107° steam to bronzed cast iron to air at 20°	.00085	.00586	171.0	.00091	K2
100° steam to iron to air at 20°	.00119	.00826	121.0	.00128	K3
153° steam to iron to air at 20°, observed	.00128	.00887	113.0	.00137	K5
153° steam to iron to air at 20°, calculated	.00129	.00895	112.0	.00139	K5
104° steam to cast iron to air at 21°	.00139	.00965	104.0	.00150	S
175° steam to iron to air at 21°	.00143	.00992	101.0	.00154	Br
127° steam to cast iron to air at 21°	.00146	.0101	99.0	.00157	S
149° steam to cast iron to air at 21°	.00156	.0108	93.0	.00168	S

A—Arsem vacuum furnace. Resistor is a helical strip of graphite.

B—Box "Treatise on Heat." Heat radiated when air is 1° F. lower than radiating body. These values appear to be only for the heat lost by true radiation and not the total; the word "radiated," however, is often used carelessly for the total, and may perhaps mean it here.

Br—Geo. M. Brill. *Trans.* Amer. Soc. Mech. Eng., Vol. XVI, p. 827. 8" bare steam pipe.

H—Carl Hering. *Trans.* Amer. Electrochem. Soc., Vol. 17, p. 164. Water cooling of brass terminals of electrodes. Brass temp. about 143°, water temperature ordinary; no steam generated.

I—Isherwood, 1867. Kent, 8th ed., p. 561. From steam at from 220° to 320° F., to water at 212° F. through pots of different metals. Rate was the same for different thicknesses of the same metal.

K1—"Kent Mechan. Eng. Pocket Book," 8th Ed., p. 855. Average practice in steam boilers is 3 lbs. of water evaporated at and from 212° F. per square foot per hour.

K2—Do., p. 656. Ordinary bronzed cast iron radiating surfaces for heating rooms give out about 250 heat units per hour per square foot with steam pressures of 3 to 5 lbs. and about 0.6 of this for ordinary hot water heating.

K3—Do., p. 571, 1st Ed., p. 477. General practice in steam heat radiators is to allow for 1.8 to 3. B.t.u. per hour per square foot, per Fahrenheit degree difference. Many writers give 2 to 2.25. The latter figure is here assumed.

K4—Do., p. 563. Numerous values for steam condensers for different temperatures and velocities of water. The values here used are 500 B.t.u. per hour per square foot per Fahrenheit degree for vertical and 730 for horizontal tubes; these are only crude averages, as the variations are considerable.

K5—Do., p. 571, or *The Locomotive*, Sept., Oct., 1892, p. 153. Total heat loss of a 2 11/32-in. steam pipe calculated from laws and tables given for the radiation and convection losses, together with correction factors, 357.1 B.t.u. per hour per foot of pipe. Observed by Ordway, 358.7. Presumably good tables covering wide ranges.

K6—Do., p. 567. As an average, 400-600 units of heat are transmitted from water to water through iron plates, per Fahrenheit degree per square foot, per hour.

K7—Do., p. 567. Heat transmitted from water to air or other dry gas through iron plates, 2 to 5 units per Fahrenheit degree, according to whether the air is at rest or moves.

K8—Do., p. 567. In a locomotive boiler, including radiant heat, 17 units were transmitted through the plates, per Fahrenheit degree per square foot per hour.

L—Irving Langmuir, *Trans.* Am. Electrochem. Soc., Vol. XX. Fine tungsten wires in flowing hydrogen. Apparently very good data. Numerous other values are given.

M—Dr. A. M. Mayer. Kent., p. 552. Relative "radiations" of heat (presumably the total heat) from different iron surfaces.

Mr—G. S. Merrill. New Metallic Filament Lamps. *Jour.* Frank. Inst., April, 1911, p. 391.

P—Peclet. "Supplee's Mech. Eng.," 2d Ed., p. 497. Termed "radiation" but presumably means total transmission.

S—F. Schumann. "Supplee's Mechan. Eng.," 2d Ed., p. 499. A table of numerous values for different temperatures from 80° F. to 300°, for steam and water; losses by radiation and by convection, in still and in moving air. They are evidently calculated values. They are for cast iron pipes or radiators, with air temperature 70° F. The losses by convection in moving air (presumably referring to the flow in the indirect heating of buildings) are 167% of those in "quiet air," hence 2/3 greater. Those by radiation are approximately equal to those by convection in quiet air. The few figures here reduced are for the total and for "moving air."

Sn—F. T. Snyder. *Trans.* Am. Electrochem. Soc., Vol. XVIII, p. 239. Curves of values calculated from Stefan's law, Peclet's constants and Richards' formula; they include both radiation and convection. The values here chosen are those marked on the curve.

St—H. G. Stott. *Power*, 1902. A 2" bare pipe heated electrically to 188°. As the heat was generated in the pipe there was no thermal resistance in flowing into the pipe, but merely from the pipe to the air; hence probably a better figure for furnace coverings than those obtained when the pipe was heated by steam, which involves the high contact resistance of steam to pipe.

WD—Winne and Dantsizen. *Trans.* Amer. Electrochem. Soc., Vol. XX. Small furnace with wire wound around tube. The total surface of the wire was used here in deducing these figures; also the furnace temperature; that of the wire was probably somewhat higher.

X—Tests made for the writer in a university laboratory with a narrow band of nickel heated electrically, the temperature being measured with a thermo couple.

Y—Common rule for heating of dynamos, namely that the temperature rise in centigrade degrees equals 100 divided by the square inches per watt. For a rise of 40° C. this gives 2.5 sq. in. per watt.

Z—Common practice in resistances embedded in enamel on cast iron, counting one side only. One watt per square inch for every 120° F.

Dimensions of Open-Hearth Furnaces Based on Practical Data.

An interesting paper by Prof. M. A. Pawloff, of St. Petersburgh, Russia, in *Stahl und Eisen*, 1911, page 1183, gives data on dimensions of open-hearth furnaces from modern European practice.

I.—Ordinary Gas Furnaces.

The dimensions are determined by: (1) the distance between the bulkheads $= L$, (2) the distance of the long sidewalls $= E$, and (3) the height of the roof above the bottom $= h_o$. $L \times E$ may be called the bath surface S.

1. *Hearth.*—Usually based on the weight of the charge T. The ratio of bath surface S to weight of charge T is not constant in the different furnaces, but decreases regularly with increasing capacity of the furnace. This ratio depends upon the depth of the bath (in Europe not over 300 mm, or 11⅞ in., and often less) and the furnace design. Table 1 gives some ratios of modern basic open-hearth furnaces:

TABLE I.

Charge T Tons.	$\frac{S}{T}$ in Square Feet per Ton.	Bath Surface S, Square Feet.
12.5	12.9	161.5
15.	12.1	181.7
17.5	11.3	191.8
20.	10.76	215.3
25.	10.22	258.3
30.	9.68	290.6
40.	8.87	355.1
50.	8.61	430.4
60.	8.07	484.2

Sometimes we find $\frac{S}{T}$ somewhat greater, but rarely below above values. If the ratio $\frac{S}{T}$ has been well chosen, the most favorable value for T can be found only by practical experience under running conditions.

2. *Dimensions of the Furnace.*—The dimensions of the furnace are derived from the hearth. The width of the furnace in Europe is not over 11.5 ft., but in America it is as high as 14.8 ft.; the greatest length is 43 ft. The length of the furnace L may be found from the ratio $m = \frac{L}{E}$ after S has been chosen from Table 1. In general m is between 2 and 3; in Europe usually 2½ or 2¾. For larger furnaces $m = 3$ is recommended. Therefore,

$$E = \frac{L}{m} \text{ and } S = L \times E = L \times \frac{L}{m}$$

$$L = \sqrt{S\,m} \text{ and } E = \frac{S}{L}$$

TABLE II.

Relations Between Capacity, Length and Width of Open-Hearth Furnaces

T in Tons.	$\frac{L}{E} = m$	L Feet.	E Feet.	S Square Feet.
30	2.66	27.9	10.5	290.5
	2.70	28.	10.3	
40	2.70	31.	11.5	355.
	2.75	31.3	11.3	
50	2.75	34.4	12.5	430.
	2.80	36.1	11.9	
60	2.80	36.8	13.1	484.
	3.00	38.1	12.7	

3. *The slope toward the tapping hole* is greatest in the middle of the hearth, being 1/12 to 1/16 of the width, in average 0.07, which corresponds to the tangens of an angle of 4°.

4. *Furnace Volume.*—The heights h_o above the bath for 40-50-ton furnaces should not exceed 83 in. It depends largely upon the gas quality and quantity and the process in use for making steel. Only local conditions and furnace practice will decide this question, but it can be said that oil-fired furnaces require a greater distance than gas-fired furnaces.

5. *Bulkheads.*—The air inlet area is usually 23.25-38.75 sq. in. per ton of charged material.

The gas inlet area is 1⅓-2½ times smaller than above values.

In furnaces of the same system the area of the gas inlet decreases regularly with increasing furnace capacity.

	10-Tons Square Inch.	20-Tons Square Inch.	30-Tons Square Inch.	45-Tons Square Inch.	150-Ton Mixer Square Inch.
Air inlet	47.4	38.1	30.2	27.9	12.9
Gas inlet	39.1	29.8	23.6	18.6	7.8
Ratio	1.214	1.281	1.283	1.5	1.66

The reason for this is that the drop in gas pressure, caused by friction, is inversely proportional to the ratio of circumference to area.

Since the ratio $\frac{S}{T}$ decreases with increasing capacity, we may also compute the area of inlets from the ratio of this inlet area to bath surface. The author found that this ratio has an almost constant value, being 3.24 sq. in. for air inlet per square foot (or 225 sq. cm per square meter).

6. Assuming the width of the inlets, the height is found from

above data. The dimensions of the gas inlets are derived from those of the air openings. The author suggests that the ratio of the cross-sections of the air inlets and gas inlets should be proportional to the quantities of heat necessary for heating air and gas or to the weights of the respective checkers.

7. *Gas and Air Flues.*—The slope of the gas flues (inlets) varies between 12° and 18°, and that of the air flues between 30° and 40°, and depends largely upon the size of the furnace. In American furnaces the gas burns almost completely, although only a single broad opening forms the gas inlet.

8. *Vertical Flues.*—Vertical flues have 25 per cent greater area than their respective inlets for the larger furnaces, and 50 per cent greater area for the smaller (20-25-ton) furnaces.

9. *Length of Bulkheads.*—The length of bulkheads depends upon the type of the different furnaces. The distance between vertical gas flues and gas inlets is between 9.8 ft. and 13.1 ft.; for air flues the distance is between 4.9 ft. and 8.2 ft.

Regenerators.

10. *Volume.*—Modern practice per ton charge is 176.6 cu. ft. checkers for small European furnaces and 123.6 cu. ft. for the largest American furnaces. In economically working furnaces, which have a low temperature drop (about 120° C. or 216° Fahr. per hour) between two reversals, the author finds that there are 90 to 110 lb. checkers per pound of coal consumed per hour.

The author recommends to base the design of the checkers on the bath surface. By using the above values of the ratio $S \div T$, the checker volume per unit of hearth surface is almost constant = 4.5 cu. m per square meter = 14.76 cu. ft. per square foot.

11. *Dimensions of Regenerators.*—The checkers in large furnaces are 80 per cent; in smaller furnaces, 70 per cent of the entire chamber volume. The depth of the checkers should not be less than 13 ft., and if possible 16 ft. The ratio between air and gas chambers varies between 2 and 1. For cool producer gas it is 1.5 and for hot gas 2; 1¾ being a fair average ratio.

Chimney Flues and Reversal Apparatus.

These dimensions are often too small, because in the case of rebuilding a furnace these parts are usually left unchanged.

12. In satisfactorily working furnaces the smallest area of air flue per 100 sq. ft. bath surface is 2 to 3 sq. ft.; the area in the reversal valves should be at least 3 sq. ft. per 100 sq. ft. hearth surface.

13. Instead of giving air and gas valve the same dimensions it is better to make the ratio of their cross-sections equal to the ratio of the cross-sections of their flues or to the ratio of their respective checker volumes.

14. *Flues Between Valves and Chambers.*—Flues between valves and chambers are made with twice the cross-section of the valve flue cross-section. Thus, for the air flue a cross-section of about 6 sq. ft. is chosen per 100 sq. ft. hearth surface.

15. *Flues Between Valves and Stack.*—The cross-sections of the flues between valves and stack should equal the sum of the cross-sections of the two flues which run to the valves of one side.

16. The same area has to be given to the stack at its base. The opening at the top of the stack is $1/20\,S$ where S is the bath surface, making the diameter $d = \frac{\sqrt{S}}{4}$

The height of stacks should be 131 to 148 ft. for larger open-hearth furnaces.

II.—Oil-Fired Furnaces.

The ratio $\frac{T}{S}$ remains the same. The distance between roof and bath has to be greater to give the oil time for complete combustion; for 20-25-ton furnaces this height should be 71-79 in. The air inlet should be 4 sq. ft. per 100 sq. ft. hearth surface.

The volume of the air checkers should be 11.5 cu. ft. per square foot hearth surface, and on account of the high heat value about 140 to 160 lb. of checker should be used per pound of coal gasified per hour.

The cross-section of the flue to stack should be about 8.5 sq. ft. per 100 sq. ft. hearth surface. The valve area and the opening of the stack at the top should be 4.25 sq. ft. per 100 sq. ft. surface.

III.—Mixers and Furnaces for Continuous Operations.

The above data cannot be applied to ordinary mixers, but only to such in which a part of the compounds of pig iron are to be subjected to an oxidizing treatment. However, in this case we have here a greater depth of the bath, therefore all dimensions should be based on the surface and not on the weight of the charge.

In this connection reference may be had to some tables of dimensions of European and American open-hearth furnaces published in *Stahl und Eisen* Jan. 12, 1910.

Notes on Chemistry and Metallurgy in Great Britain.

(From our Special Correspondent.)

The Corrosion of Metals.

If the conclusions arrived at by Messrs. W. R. DUNSTAN and J. R. HILL in their papers recently read before the Chemical Society (Journ. Chem. Soc., 1911; pp. 1835-1866) on "The Aerial Oxidation (Rusting) of Metals" and "The Passivity of Iron and Certain Other Metals" are to be accepted, then both the carbonic acid theory and the electrolytic theory of corrosion are untenable.

The authors had previously maintained (Trans. Chem. Soc., 1905, 87, 1548) that the presence of carbonic or any other acid is not essential to the rustings of iron and that the necessary conditions are only iron and dissolved oxygen—a view confirmed by Tilden (Trans. Chem. Soc., 1908, 93, 1356)—and held that the rusting of iron involves the intermediate formation of hydrogen peroxide, just as with other metals; but they now find that there are some substances which decompose hydrogen peroxide, but do not inhibit rusting, and therefore admit that this explanation is not entirely satisfactory or complete.

The advocates of the electrolytic theory explain the fact that pure iron may be kept immersed in pure water for months without change by assuming polarization of the iron by the hydrogen liberated as a result of ferrous ions and hydroxyl ions forming ferrous hydroxide; oxygen would then act as a depolarizer and the process would continue.

The authors object that iron would not become permanently polarized until the water were saturated with hydrogen; that iron can remain in boiling water for long periods without oxidation; and the retention of a film of hydrogen in boiling water is hardly an admissible supposition; that iron cannot facilitate the direct oxidation of the hydrogen by dissolved oxygen because the film of hydrogen keeps the metal from contact with the water; that if dissolved oxygen can depolarize iron, other depolarizers, such as potassium bichromate, chlorate and nitrate should do the same, but they do not; that the electrolytic theory does not explain the action of inhibiting agents other than the alkalis, viz., potassium chromate, bichromate, iodate, ferrocyanide and hydrogen peroxide, which all inhibit rusting, whilst the electrolytic theory demands that most of them should assist rusting by depolarizing the iron.

The authors determined the question of the solubility of iron in pure water with a negative result, and made many experiments on the cause of the action of inhibiting agents, the passivity induced by such agents, passivity and rusting, the

comparison of the oxidation of iron with that of other metals, and the formation of hydrogen peroxide during the rusting of iron.

Their conclusions from the investigations described in the first paper are: That existing theories do not adequately explain how inhibiting agents prevent rusting; but the fact that all these agents make iron passive and that the passivity persists after removal from the solution which caused it, does afford a satisfactory explanation.

The mechanism of the process of rusting of iron presents no essential difference from the aërial oxidation of other metals; traces of hydrogen peroxide are formed when iron rusts just as in the case of other metals, but the hydrogen peroxide is more probably the result of a secondary reaction than an intermediate product of the main reaction, since oxidation can apparently continue without its formation.

The aërial oxidation of metals results from the direct action of dissolved oxygen. In the case of iron ferrous hydroxide is first formed and is further oxidized to the ferric state. Iron does not pass into solution before rust is formed. The action of carbonic or any other acids is independent of the main process of rusting, which proceeds in their absence. The fact that iron which has been treated with inhibiting agents rusts when exposed to carbonic acid is due to the destruction of passivity by the carbon dioxide.

The Nature of the Passive Condition.

In the second paper the same authors give details of experiments on the production and destruction of the passive state in iron, zinc, magnesium, copper and lead. In the case of iron the theory that the passivity is due to a gaseous film was disproved by heating in a vacuum to 400° C., when the passive condition was not affected; but on heating in an atmosphere of hydrogen at 250° C., the temperature at which the magnetic oxide of iron is reduced, the passive condition was removed. This goes far to prove the correctness of the old view that the passive state of iron is due to a film of ferrosoferric oxide.

The authors find a difficulty in explaining the action of alkalis in causing passiveness, and yet state that it is known that under certain conditions iron can be oxidized by alkalis with production of a visible film of oxide. In the case of copper, the protective film has a red shade and probably consists of cuprous oxide. The film is non-metallic because it resists amalgamation of the copper by mercury.

The Value of Mechanical Tests.

Mr. C. E. Stromeyer, chief engineer of the Manchester Steam Users' Association, in his annual memorandum, gives some interesting details of experiments carried out on samples of fractured boiler plates. The results of the applied mechanical tests lead him to the conclusion that, except in cases of bad heat-treatment, none of these tests will distinguish between reliable and unreliable steel, and that chemical analysis, particularly the determination of phosphorus and nitrogen, affords the only available index when the nature of the raw material and the process of manufacture are not known. He expects that engineers will, nevertheless, not be inclined to rely much on chemical analysis which they cannot check, but will continue to prefer procuring boiler plates from makers of known reputation and applying the usual mechanical tests.

In view of the fact that the micro-structure often discriminates between good and bad steels of almost identical chemical composition, the author's statement that chemical analysis is the only guide is not likely to find a wide acceptance.

The Action of Caustic Soda on Boiler Plates.

In the same publication Mr. Stromeyer calls attention to the deleterious action of caustic soda solution on mild steel boiler plates when the density of the liquor exceeds 9 deg. Twaddell (sp. gr. 1.045, corresponding to about 4 per cent of NaHO at 15° C.) and the steel is in a state of tension. He advises that the concentration of caustic liquid in closed boilers under pressure should be avoided except for very weak solutions.

The Deflocculation of Clay.

Dr. E. G. Acheson presented a paper on "Deflocculation" before the London Section of the Society of Chemical Industry, which was received with very great interest, not only because of the desire to hear the distinguished author, but also of the interest of the subject itself. Dr. Acheson said that the unsuitability of American clays for binding material for graphite caused him to investigate the nature of the difference between sedimentary clays, such as those imported from Europe, and residual clays. After trying the effect of various solutions of organic substances he found that gallotannic acid, in dilute solution, materially reduced the quantity of water required to produce a given degree of fluidity with the clay and increased the tensile strength and plasticity. Investigation showed that the clay abstracted tannin from the solution. Briquets made from the treated clay had a strength more than 200 per cent greater than that of briquets of the original clay. Aqueous extract of straw had the same effect as gallo-tannic acid solution. Microscopical examination proved that clays treated with these substances became deflocculated, and the method of deflocculation was applied to graphite with the result that two new lubricants, "aquadag" and "oildag," were produced, consisting of deflocculated Acheson graphite suspended in water and oil respectively. The deflocculated graphite was found to be in a truly colloidal state, and the size of the particles was estimated at 75 millionths of a millimeter, or 3 millionths of an inch. Acids, alkalis and salts caused aggregation of suspended deflocculated particles; and this might explain the formation of mud-banks at the mouths of rivers.

The Volume Relations of White and Grey Cast Iron.

The well-known superiority of grey iron castings compared with those of white iron has led E. Rosenberg (*Stahl. und Eisen*, 1911, 31, 1408) to investigate the relations between the volumes of the two kinds of iron on cooling from the liquid state to 1000° C. and at 0° C. Two methods of calculation were employed and gave concordant results showing that the volume of the grey iron expanded on solidification, whereas the volume of the white iron diminished, and that the expansion of grey iron increased with the percentage of graphite. Two specimens of grey iron both had a sp. gr. of 6.89 when liquid, 6.75 and 6.46 at 1000° C., and both 7.1 at 0° C. Two specimens of white iron also had a sp. gr. of 6.89 in the liquid state, 7.22 and 6.91 at 1000° C., and both 7.6 at 0° C. Pig iron with 3 per cent graphite and 1 per cent combined carbon gave for sp. gr. 7.1 liquid, 6.9 at 1000° C. and 7.2 at 0° C.; another pig with 1 per cent graphite and 2 per cent combined carbon had a sp. gr. of 7.1 liquid, 7.2 at 1000° C. and 7.4 at 0° C. The calculation for the last two sets of figures was based on the relative densities of pure iron, iron carbide and graphite, and the separation of graphite from carbide during solidification.

Engineering Imports and Exports.

The Board of Trade returns for the ten months ended on the 31st of October give the value of imports of iron and steel, including manufactured articles, as £9,109,220, an increase of £1,811,580 compared with the corresponding period of 1910; and exports £35,804,215, an increase of £132,763. Imports of other metals, including manufactures, were £22,976,488, a rise of £3,070,412; with exports £9,070,302, also a rise of £491,199. In electrical goods the imports reached £1,143,892 and the exports £2,286,868, both showing decreases of £167,287 and £994,036 respectively. Imports of machinery went up to £4,842,529, and the exports to £25,444,377, giving increases of £1,144,098 and £1,259,027 respectively. Imports of new ships are put at £60,993, and exports at £4,413,072; the imports showing an increase of £38,266, but exports are lower by £3,631,457. The figures for the month of October show that a satisfactory recovery is in progress from the depression caused by the recent labor

troubles; and, moreover, that the values of exports show considerable improvement on those for October of last year.

Market Prices.
November, 1911.

Copper has shown an upward tendency. Starting at £55.10.0 it has risen steadily throughout the month, closing at £58.17.6.

Tin has been, on the whole, ascending in price. Starting at £189 it dipped to £186.5.0 by the 3d, but afterwards rose to £198.5.0 (20th), and closes at £201.10.0.

Lead has risen a little. Starting at £16 it reached £16.2.6 on the 21st and closes at £16.3.9.

Haematite started rising decidedly on the 10th. Opening at 60/10, it rose to 61/10 on the 10th and 62/6 by the 21st, closing at 63/—.

Scotch Pig has been steadily increasing, with slight dip (6d) on the 17th. It opened at 52/3, rose to 53/4 by the 16th and 54/— by the 21st. It closes, however, at 53/9.

Cleveland has risen steadily, starting at 46/2, it reached 48/— by the 21st, and closes at 47/9.

India-rubber has remained steady. Opening at 4/3, it was at 4/3½ on the 8th and 4/3 on the 22d; closes at 4/4.

Alum, lump, loose, per ton	£5.12.6
Antimony, black sulphide powder, per ton	20. 0.0
Borax, British refined crystal, per ton	16. 0.0
Copper ore, 10 to 25 per cent, per unit	9/7 to 10.1½
Copper sulphate, per ton	20.15.0
Carbolic acid, liquid, 97/99 per cent, per gal	1.7
Camphor, 1 oz. tablets	1.9½
Caustic soda, ash 48 per cent, ordinary, per ton	5.10.0
Creosote, ordinary good liquid, per gal	3
Hydrochloric acid, per cwt	5.0
Mica, small slabs, original cases, per lb	6d to 2.0
Naphtha solvent, 90 per cent, 160° C., per gal	9
Platinum, per oz., nominal	9. 5.0
Petroleum, Russian spot	5¾
Quicksilver, per bottle	8.10.0
Sal ammoniac, lump, first, del. U. K., per ton	42. 0.0
Sulphate of ammonia, f.o.b. Liverpool, per ton	14. 8.9
Sulphur, recovered, per ton	5. 0.0
Shellac, Standard T. N. Orange Spots, per cwt	4. 4.0
Tin ore, 70 per cent, per ton	£124 to 126. 0.0
Zinc, Vielle Montagne, sheets	30. 2.6
Increase.	
Copper ore, per unit	£0. 0.3
Copper Sulphate, per ton	15.0
Naphtha, solvent, per gal	1
Ammonia sulphate, per ton	1.9
Tin ore, per unit	2. 0.0
Petroleum, per gal.	0¾
Decrease.	
Antimony sulphide	£2. 0.0

Electrolytic Zinc Process in California.—From a statement of the Reed Zinc Co. of San Francisco we gather the following: "The Reed Zinc Co. is the result of the efforts of Frank A. Leach, former Director of the Mint, and his son E. R. Leach, the melter and refiner of the U. S. Mint at San Francisco." A site on the southwestern shores of the Bay of San Francisco has been chosen with ample railroad facilities. "The process by which the company proposes to handle ores is covered by patents of C. J. Reed and Carl Hering, and briefly stated consists of finely grinding and roasting the ore in a specially designed furnace, thus converting the metallic sulphides to a mixture of oxides and sulphates which are subsequently brought into solution. The zinc sulphate is separated, and the zinc reduced to metallic form by electrolytic methods. In subsequent procedure sulphuric acid and other valuable byproducts are recovered. . . . In the roasting process liquid sulphur dioxide and sulphur will be recovered." Mr. Reed is widely known as one of the founders of the American Electrochemical Society. We reserve discussion for our next issue.

Synopsis of Metallurgical and Chemical Literature

Iron and Steel.

Coke-Oven Gas and Tar Open-Hearth Furnaces.—In a discussion of a paper read by Dr. Ing. Petersen, Düsseldorf, at the meeting of German Steel & Iron Masters (*Stahl und Eisen*, 1910, p. 80), Mr. Maltitz made the following remarks. He wondered whether the Hubertushütte was burning coke-oven gas in open-hearth furnaces successfully. He personally made tests along this line in the Dominion Iron & Steel Co., but with negative result. It was impossible to preheat the gas in regenerators, so the gas was burned directly like natural gas, flowing through 6" pipes. The heating effect was very low, therefore tar was also sprayed into the furnace by nozzles. The result of burning tar was not successsful because the furnace roofs stood only 90 heats.

Mr. Amende of Hubertushütte replied that they had used coke-oven gas for two and a half years with great success, sometimes as much as 70 per cent of the coal being replaced by coke-oven gas. The durability of bulkheads and roofs decreased 8 to 10 per cent, but that of the checkers increased about 50 per cent. They ran 550-600 heats without repair work on bulkheads and roofs and as high as 1050 heats on the same checkers. The steel quality was not effected at all. In twenty-four hours, 1,800,000 to 2,600,000 cu. ft. of coke-oven gas were consumed, including iron and steel foundry's heating molds. The coal consumption per ton of steel was reduced from 31.8 per cent (1906) to 14.9 per cent (1909) by the use of coke-oven gas. A blast-furnace gas main is under construction, and it is hoped to further reduce the coal consumption by this means. The coking coal is of poor quality, so that only 45 per cent gas is available. The coke-oven plant contains ninety Otto-Hoffmann ovens with a capacity of 320 tons coal per twenty-four hours. The coke-oven gas analysis is: 6.5 CO_2, 1.2 O_2, 10.4 CO, 38.7 H_2, 16.4 CH_4, 24.8 N_2, b.t.u. = 377.

Different Types of Mixers.—The following details of various European mixers should be of interest: On the Julienhütte in Upper Silesia (*Stahl und Eisen*, 1910, page 2147) there is a 150-ton mixer, electrically tipped on rolls for six open-hearth furnaces at 40 tons each. The pig-iron inlet and slag-tapping opening are on one side, the tapping opening for pig iron on the other. The mixer is heated by producer gas, but has only air regenerators; the temperature inside is 2370° to 2550° Fahr. Coal consumption for 400-500 tons pig iron in twenty-four hours = 8 tons. The bottom is lined with magnesite.

On the Georgs-Marienhütte in Osnabrück (*Stahl und Eisen*, 1910, page 2151), there is one mixer for 150 tons and one for 250 tons for five open-hearth furnaces at 40 tons each. Pig iron is pre-oxidized in the mixer and this metal charged in the open-hearth furnaces. The mixer is on rolls; the bulkheads can be moved; all is done by hydraulic pressure. The gas for the five open-hearth furnaces is produced in nine Kerpely producers. The mixer is heated by blast-furnace gas from a 35.4-in. pipe line. The burning of blast-furnace gas is considered a secret, but it can be said that this gas burns with a high temperature. Blast-furnace gas is also used in the soaking pits and for some time it was used in open-hearth furnaces, but this was abandoned on account of the insufficient amount of gas. On account of pre-oxidizing in the mixer, the time for a heat in the open-hearth furnaces is considerably reduced, and two open-hearth furnaces now produce at least as much steel as three furnaces without mixer.

At Donawitz, Styria (*Stahl und Eisen*, 1910, page 21), there is a 150-ton mixer for twelve open-hearth furnaces of 30 tons each, heated by producer gas. There are twenty-five Kerpely producers furnishing gas to the open-hearth plant, to the ore-roasting and mill furnaces. Producer gas analysis: CO_2 = 2-2.5 per cent; CO = 30-31 per cent; H_2 = 12-14 per cent; CH_4 = 1 per cent. The bulkheads and roofs stood 583 heats; the checkers 3000-3600 (in 1908). Monthly production per furnace 3200 to 3400 tons.

Gold and Silver.

Available Calcium Oxide in Lime for Cyanide Process.—The almost universal use of lime as a means for maintaining protective alkalinity in cyanide solutions, and the fact that the percentage of calcium oxide contained determines its value, calls for quick and efficient means of estimating the quantity of calcium oxide in burned limestone. In the November, 1911, *Bulletin* of the Amer. Inst. of Min. Eng. Mr. Luther W. Bahney presents a method which he has found to give quick and accurate results. A solution of oxalic acid is made, containing 14.6068 g per liter of water, and used to titrate the alkalinity of a weighed quantity of lime. In order that 1 cc of the oxalic acid solution shall be equivalent to 1 per cent CaO, he uses 650 mg of lime for each determination. This quantity is arrived at by the following proportion, in which x equals 650:

Lime	Lime	Oxalic	Oxalic
56.09	x	126.048	1.46068

The lime sample must be ground to pass a 200-mesh screen. Into a 300-cc Erlenmeyer flask place 50 cc distilled water, and add 650 mg of the sample to be tested. Stopper the flask and shake vigorously for 10 seconds; add two drops of phenolphthalein and then run in the standard oxalic acid solution until the pink color is discharged; then replace the stopper and shake. When the color returns, if it is due to lime it will be a bright pink, and perhaps 0.5 cc more solution will be required to finish the titration. But if the returning color is a weak pink, returning slowly, it indicates magnesia, and the lime titration may be regarded as finished with the first discharge of color. The mixture in the flask should be shaken frequently during the titration. The reading of the burette is the percentage of calcium oxide in the lime.

Slime Filtration.—The physics of slime filtration has been made the subject of an exhaustive investigation by George J. Young, of Reno, Nev., his paper being published in full in the November, 1911, *Bulletin* of the Amer. Inst. of Min. Eng. After reviewing the various methods of filtering slime, and the physical condition of the material commonly called slime in the metallurgical industry, he describes experiments made to determine the effects of variations in pressure, temperature and physical composition of slime. Numerous curves and drawings accompany the paper. The author's comprehensive definition of slime is: A slime consists of a mixture of sands finer than 150 or 200-mesh screen, with an amorphous clay-like material, consisting principally of hydrated aluminium silicate.

Certain practical conclusions may be drawn:

1. The proportion of clayey material in ores which are subjected to all-sliming and filtration should be maintained at a minimum.

2. The slime pulp should be as free as possible from sands coarser than 150-mesh screen, and as large a proportion of the pulp as possible should consist of material passing a 200-mesh screen.

3. The slime pulp before filtration should be settled to as thick a consistency as possible consistent with ready handling by pumps and in pipes.

4. The temperature of the slime pulp should be maintained between 20° and 30°C. or higher.

5. The temperature of the wash water and the pulp should be the same.

6. Vacuum pressures should be varied until the proper intensity for the given slime is obtained.

7. Where very clayey slime is to be filtered, as much fine sand (limited as above stated) should be crowded into the pulp as it will carry without undue settling and clogging.

8. No. 10 canvas supported by slats gives the best all-round service for the thick cake, and No. 12 canvas on wire netting answers the requirements for the thin-cake filtering machines.

9. With slimes containing a large proportion of colloid or clayey material, pressures greater than those obtainable with the vacuum apparatus are of questionable advantage.

10. With slimes containing a large proportion of clayey material the vacuum filters should be used.

11. With slimes containing a small proportion of clayey material and much fine sand, both vacuum and pressure filters could be used with perhaps equally good results.

12. With slimes containing much coarse and fine sand the chamber filters with air agitation and high pressures would perhaps give the best results.

13. Of the vacuum filters the thin-cake continuous filters are a decided improvement over the thick-cake filters.

Parral Tank for Slime Agitation.—The Pachuca tank probably has become as widely known as any device used in the cyanide industry, and it has been accepted as few other devices have been. Some of its inherent weaknesses have been pointed out, but apparently without effect on its remarkable progress throughout the world. In view of these facts, more interest

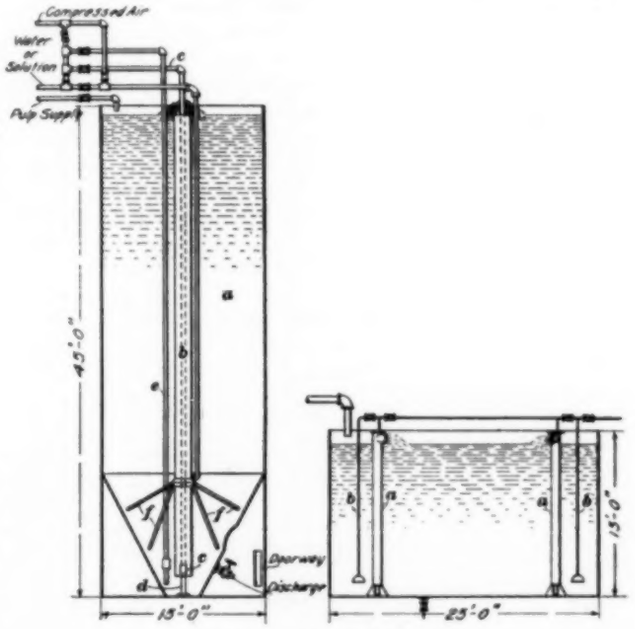


FIG. 1.—PARRAL TANK.

attaches to an invention known as the Parral tank, from its use in the Mexican district of that name. It is described by the inventor, Bernard MacDonald, of Pasadena, Cal., in a paper read before the San Francisco meeting of the American Institute of Mining Engineers. The Parral tank is in use at the mill of the Veta Colorado Mining & Smelting Company, Parral, Mexico.

The dimensional relations of the Parral and Pachuca tanks are shown in Fig. 1. It will be seen that the objectionable height of the Pachuca tank is avoided in the low form of the Parral. In the Pachuca tank a definite ratio of height to diameter must be preserved, while in the Parral no cognizance of this ratio need be taken. At the Veta Colorado mill the Parral tanks are 25 ft. in diameter and 42 ft. in height, with a capacity two and one-half times as great as the standard Pachuca tank. For agitating the slime and raising it from the flat bottom to the top of the tank four transfer pipes *a* are installed, each 12 in. in diameter and set 12 in. from the bottom and 4 ft. from the tank side and equidistant from each other. In a tank of larger dimensions more transfer pipes would be needed. Compressed air is admitted to the bottom of these transfer pipes through a nozzle fitted with a ball valve which automatically opens and closes as required in the jet feeding of compressed air. This ball nozzle is a more efficient and satisfactory device than the elastic sleeve usually used on the bottom of the air pipe in the Pachuca tanks.

On the top ends of the transfer pipes are bolted tees of equal diameter, with the run in line with the pipes and the outlets so

directed as to discharge the pulp in line of segment cords to the circumference of the tank. The discharge of all the pipes being in the same direction, the force of the discharge sets up a spiral or rotary flow in the tank charge, which in a short time extends down to the bottom of the tank. This motion carries the particles round and round so that their length of travel before reaching the bottom of the tank is greatly increased over what it would be if the settlement were in a vertical line.

When Parral tanks are being filled the inflowing pulp is kept in agitation by means of compressed air admitted through auxiliary pipes shown at *b* in the figure. When the tank is filled to within 10 ft. or 12 ft. of the top the air is closed off the auxiliary pipes and opened on the air lift pipes.

The following table gives a comparative statement of the details of Pachuca and Parral tanks.

	Pachuca	Parral
Height in feet	45	42
Diameter in feet	15	25
Area of bottom of tanks, sq. ft.	176.7	490.8
Holding capacity for each foot in height, cu. ft.	176.7	490.8
Number of air-lift tubes	1	4
Diameter of air-lift tubes, in.	16	12
Cross-sectional area of air-lift tubes, sq. in.	201	452
Diameter of air pipe in lift tubes, in.	1.5	1
Cross-sectional area of air pipes in lift tubes, sq. in.	1.7671	3.1416
Proportional area tank bottom for each sq. in. of cross-section air-lift tubes, sq. ft.	.8	1.8
Area of tank bottom for each sq. in. of compressed-air pipe, sq. ft.	100	156

Cyaniding in Australia.—Milling and cyaniding on the Charters Towers gold field in Australia has been unfavorably criticized for excessive costs, which in the past have ranged from $1.75 to $3 per ton for milling (amalgamation and concentration), and from $1.50 to $2 per ton for cyaniding. In the October, 1911, issue of the *Mining and Engineering Review*, Melbourne, Mr. W. A. MACLEOD reviews the situation and shows that by improvement in the same old methods the costs have been reduced to $1 per ton for crushing, amalgamating and concentrating, and to $0.75 per ton. The author is of the opinion that the present costs could not be sufficiently reduced by new equipment to warrant the capital expenditure which would be necessary.

At the Brilliant Extended mill the ore is crushed by Blake crushers to about 1½-in. size and fed to Huntington mills and stamps. The Huntingtons are 5 ft. in diameter, running at 75 r.p.m., and fitted with 18-mesh screens. They crush 33 tons of hard quartz in 24 hours. The mills are economical in power, but expensive in maintenance, and their cost per ton exceeds that of the stamps by nearly 12 cents. The stamps weigh 1250 lb., have a 7-in. drop 103 times per minute, and crush 4.5 to 5 tons per stamp in 24 hours. Grading analyses show that the stamps crush finer than the Huntingtons, using the same screens.

The value of the ore ranges from $11 to $12.25 per ton. The tailing assays from 50 cents to 60 cents per ton. About two-fifths of the recovery is made by amalgamation, two-fifths by concentration and one-fifth by cyanidation. The concentrates are sold to smelters, as about $4.85 per ton more is realized in this manner than by local treatment with fine grinding, amalgamation and cyanidation.

The tailings from the concentrator are cyanided by percolation in tanks. Experiments have been made looking toward the agitation and filtration of the slime, but only a 55 per cent extraction was obtainable after four to five days' agitation. The old practice in treating the tailings was to spread them out to dry in the sun for some weeks, after which they were leached in tanks. This practice was expensive on account of the labor involved and the poor mechanical arrangements. It has been improved by first classifying the sand and slime in spitzkasten and then spreading the products separately. The sand dries rapidly, and the slime bed is dried more rapidly than before by breaking it up with disk harrows. When dry, the sand and slime are again mixed and ground, and filled into tanks by belt conveyors. The time of treatment is about five to six days; the value of the charge about $2.50 to $3, and of the tailing 40 to 65 cents. The tanks are discharged by sluicing, using 100 tons of water for 200 tons of charge. Precipitation of the gold solution is by zinc shavings.

Zinc, Lead and Copper.

Progress in Zinc Smelting.—In an advance chapter of "Mineral Resources of the United States," written by Mr. C. E. SIEBENTHAL, a number of improvements attempted are briefly described. The Saeger cleaning and charging machines are in use in Silesia. Mounted upon a truck in alignment with the sloping retorts are banks of screw conveyors. There are as many banks of conveyors as there are banks of retorts, and each bank contains from three to eight conveyers. After the condensers are removed, the cleaning machine withdraws the charge by inserting the screws into the retorts. Then follows the charging machine, which is similar in character, and which forces the charge into the retorts by means of screw conveyors. The advantages are: Saving of half the labor charge; shortening of the injurious cooling-off period while cleaning and charging; lengthening the time for distillation, and more healthful conditions of work.

Two forms of moving furnaces have been designed. The Wettengel rotatable furnace was described in this journal for April, 1911, page 198. The Hughes revolving furnace consists of a rotary retort contained in a rotary combustion chamber. The charge enters continuously at the upper end of the gently sloping retort, which discharges both residue and zinc vapor into a stationary cylindrical condenser at the lower end. The waste heat of condensation is used to preheat the charge. A retort 8 in. in diameter and 7 ft. long has a capacity of about 5 tons of roasted zinc concentrate in 24 hours. A commercial-sized retort is installed at the plant of the Hughes-West Continuous Zinc Distillation Company, Southampton, near St. Louis, Mo.

The Hopkins process of smelting zinc-lead concentrate in ordinary retorts consists in the use of a charcoal or carbon filter between the retort and condenser. Higher recovery and better grade of spelter are the improvements claimed for this process. In a nineteen weeks' comparative test at Swansea between ordinary condensers and the Hopkins condenser the latter gave 7 per cent greater recovery.

Sulman and Picard have patented a method of smelting very fine zinc-lead concentrate and slime by briquetting the roasted material with bituminous coal and pitch in the following proportions: Roasted ore, 70 per cent; bituminous coal, 25.5 per cent; pitch, 4.5 per cent. After retorting the briquets they are found to remain intact in a condition suitable for direct charging into a lead blast furnace. Experiments indicate that a recovery of 80 per cent of the zinc can be effected from this material, and that the retorts have an increased capacity of one-third, with a saving of time in charging and cleaning.

Electrostatic Separation.—At the mill of the Calumet & Sonora of Cananea Mining Company, near Cananea, Mexico, Huff electrostatic separators are in operation on zinc-copper concentration middlings. The details of operation are given by Mr. J. N. HOUSER, superintendent, in the December, 1911, issue of the *Mexican Mining Journal*. The middling from the concentrator assays about as follows: Zn, 30 per cent; Cu, 6.5 per cent; Pb, 3 per cent; Fe, 10 per cent; Ag, 3 oz. per ton. The middling is dried and sized into three grades, 2 mm to 20-mesh, 20 to 40-mesh, and minus 40-mesh. The coarse grade is treated on a Huff standard machine, and the two finer grades on modifications known as the toboggan type, which was developed at the plant of the American Zinc Ore Separating Company, Platteville, Wis. The machines are charged with static electricity at

20,000 volts. The mill is now making a zinc concentrate assaying Zn, 52 per cent; Fe, 4.5 per cent; Cu, 0.8 per cent. A copper product also is obtained assaying Cu, 16 per cent; Fe, 22 per cent; Zn, 10 per cent; Ag, 10 oz. per ton.

A part of the material treated is from an old dump of accumulated middling which, by exposure to the elements, has caused a coating of copper to be deposited on the zinc, thereby destroying the natural difference in conductivity. In order to restore this difference, the middling is given a bath of cyanide solution which dissolves the copper film. It is then dried and treated as usual. The plant is handling 35 tons per 24 hours, and is run wih three men per shift. The cost is $1.20 per ton treated, but this cost will be reduced when fresh mill middling only is handled.

Recent Metallurgical Patents.

Iron and Steel.

Eliminating Occluded Slag.—Molten steel or iron, by whatever process prepared, usually contains small particles or globules of slag which, unless they be removed, impair the quality of the metal. For the purpose of accelerating the removal of these bodies, Mr. ROBERT LOCK, of Tarentum, Pa., proposes to tap the fluid metal from the furnace or converter into a ladle in the usual way, the slag being allowed to remain on top of the molten metal. From this ladle the metal is transferred in a thin stream to a second ladle, preferably by bottom pouring so that the supernatant slag cannot be included. At the same time varying amounts of lime or other reagent, preheated or not, are gradually added with the stream flowing into the second ladle. By thus causing the metal to flow in a thin stream, and to come in contact with the lime, the globules of slag are eliminated from the metal. The inventor claims that metal produced by this method has a high degree of purity and is less susceptible to electrolytic dissociation. (1,008,420, Nov. 14, 1911.)

Process of Treating Coke.—Owing to the presence of sulphur in coke usually used in melting iron ore or in melting iron in cupola furnaces, the molten product becomes contaminated with a percentage of sulphur which must subsequently be removed. To prevent the sulphur in the coke from uniting with the iron Mr. HARRY SPURPIER of Detroit, Mich., proposes to treat the coke with a substance with which the sulphur will more readily unite. Sodium carbonate is preferred as being a cheap substance soluble in water. He recommends that a solution of this salt be used to impregnate coke prior to its use in the furnace. When the coke is used a soda salt of sulphur will be formed as a slag which will contain not only the sulphur in the coke, but some of that in the original iron as well. Other salts of potassium and sodium can be used. (1,007,153, Oct. 31, 1911.)

Gold and Silver.

Slime Thickener and Decanter.—In the treatment of slime in the cyanide process, some means of thickening the pulp and decanting clear solution is quite essential. Fig. 1 illustrates a form of thickener designed by Mr. MARK R. LAMB, of Milwaukee, Wis., similar in some respects to the design of the Dorr machine. It consists primarily of a flat-bottom tank 15, with an inlet for slime pulp at 1, and a circular launder 3 and spout 7 for the discharge of clear solution. The device for drawing thickening pulp to the central bottom of the tank consists of four radial arms 12 provided with blades 14. These are made to revolve by means of a driving mechanism placed beneath the tank, in which a worm 20 drives a gear wheel 19 attached to a vertical shaft 91 on which is keyed a frame 17 extending upward into the tank. The frame is perforated at 16, just inside the tank, for the purpose of allowing the thickened pulp to enter the bottom of the pipe 6 which is connected with a diaphragm pump 4, which removes the thickened slime as fast as the revolving arms and blades deliver it to the center of the tank. Provision is made to raise the arms 12 so that their position may be regulated with respect to the thickness of the layer of material at the bottom of the tank. (1,007,954, Nov. 7, 1911.)

A device similar in purpose to other thickening machines, but having in combination means for elevating the thickened slime or washed sand to another tank for further treatment, is shown in Figs. 2 and 3. Fig. 2 is a part sectional elevation of the invention, showing a tank and attached elevating means. Fig. 3 is a view of the screw conveyor as seen from above and at right angles to Fig. 1. The invention is that of ALEXANDER

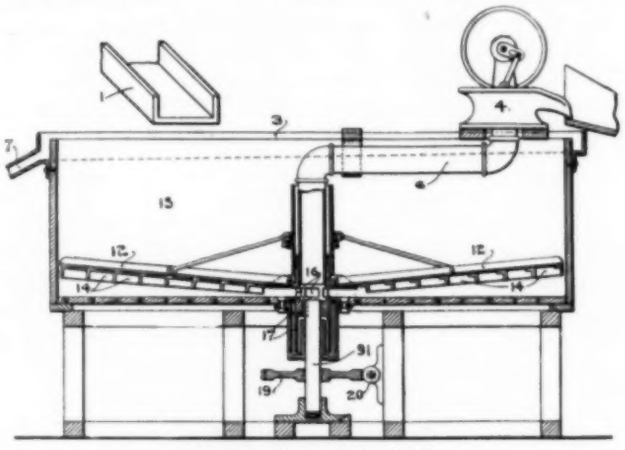
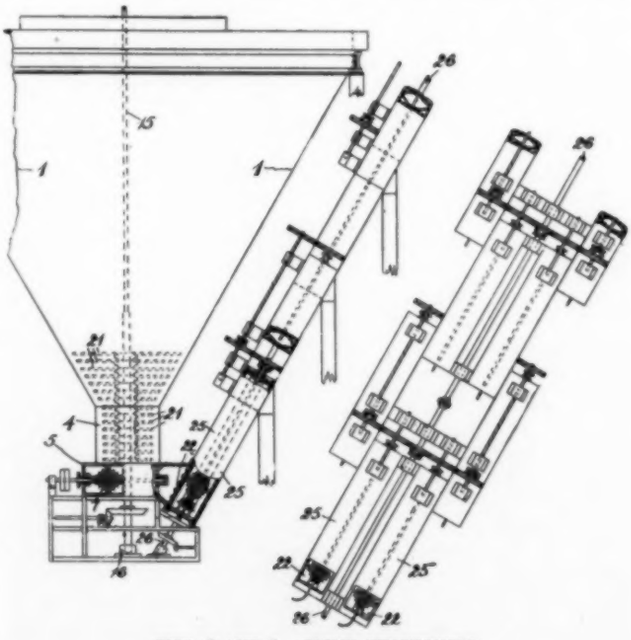


FIG. 1.—SLIME THICKENER.

J. ARBUCKLE, of Johannesburg, Transvaal, South Africa. Referring to the figures, 1 represents a settling tank having a lower cylindrical portion 4 connected with a box 5 in which intermittent or continuous helical feed conveyors 7 are placed. By means of openings 22 in the frame between the box 5 and the inclined screw conveyors 25, thickened pulp can pass to the



FIGS. 2 AND 3.—SLIME THICKENERS.

latter for elevation. In the tank 1 is a vertical shaft 15 running in a step bearing 16, with radial arms 21 attached in the form of an intermittent helix or screw thread. In operation these radial arms force the slime or sand downward into the box 5, whence it is driven by the screw 7 into the inclined screw conveyors 25. It will be observed that the screw conveyors are arranged in sections or lifts and that the slime delivered at the top of one section is raised in the next, and so on. One drive

shaft 26 serves for the operation of all the screw conveyors, suitable gearing being arranged as shown. The inventor's idea in arranging the inclined conveyor in sections was to avoid the difficulties in driving long screw conveyors, and to provide for intermediate supports in its length. (1,008,513, Nov. 14, 1911.)

Electrolytic Amalgamation.—For the recovery of finely divided gold, silver or platinum occurring either in native sand or finely ground rock, electrolytic amalgamation has offered hope to many inventors. It has been pretty clearly demonstrated that in its application to plates electrolytic amalgamation has failed to produce any good results. With the use of mercury riffles or wells, however, it has seemed possible to secure better results. In Fig. 4 we illustrate a device of JAMES H. ALLING, of Columbia, Cal., designed to use electricity in connection with amalgamation on plates and in wells or riffles.

In the drawing, 24 represents a pipe and valve controlling a flow of sand and water or other material containing finely divided gold. 32 represents a pipe and valve from which a solution of potassium cyanide, mercuric chloride, or other similar substance can be added to the flow of pulp for the purpose of

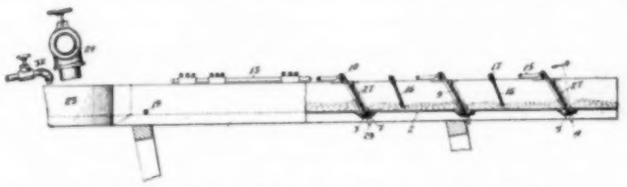


FIG. 4.—ELECTROLYTIC AMALGAMATING APPARATUS.

accelerating the amalgamating action by cleansing the gold. The sluice box is indicated at 1, and the plate at 2, with depressions or mercury wells at 3. The plate is continuous and is connected at cathode. Adjustable anode amalgamating plates are placed at 9, extending to within ⅛ in. of the surface of the mercury, and, in addition, there are swinging carbon anodes 16 at intervals above the plate.

About 4 tons of water are used per ton of sand, and the quantity of cyanide and mercury solution used is proportioned to the value and condition of the sand. The inventor has found that from ½ lb. to 1 lb. of cyanide and 2 oz. mercuric chloride in 5 gal. of water gives good results when introduced in the proportion of 1:1000 or 1:2000 parts of sand pulp. The electrolysis of the cyanide and mercury solution cleans the gold and facilitates amalgamation. A current density of 1/10 amp to 1 amp per square foot of cathode surface is used according to the value of the ore. Five volts are found to be satisfactory in practice. The inventor claims to have recovered and saved over 95 per cent of the value of the material which could not be saved by ordinary processes.

The surface of the anode is amalgamated in order to replenish the aqueous solution of mercuric chloride, and in order further to keep the anodes supplied with mercury the inventor places a plate 27 in front of each anode, a cup being formed at the top which is kept filled with mercury. As the anode plates are perforated with fine holes, the mercury finds its way through them onto the surface exposed to the flow of sand and solution. A suitable mercury trap is used at the end of the plate to catch particles of mercury that may be washed out of the sluice. (13,307, reissued Nov. 7, 1911.)

Zinc and Copper.

Preparation of Oxide of Zinc for Smelting.—The oxide of zinc obtained by precipitation from solutions of zinc salts, or from the condensation of zinc fume is usually so light and bulky that it is not regarded as a suitable substance for smelting. It is the object of an invention of Mr. R. W. E. MACIVOR and WOLDEMAR HOMMEL, of London, England, to overcome this difficulty by producing dense zinc oxide. The process consists in mixing the oxide with zinc chloride substantially in proportions to form an oxy-chloride of the formula $ZnCl_2 . 6ZnO$. This compound has the property of setting, like cement, into a dense solid, from which the zinc chloride can be volatilized at a temperature of 300° C. to 400° C., and collected for further use. The dense oxide remaining is then in suitable condition for smelting. A modification of the process consists in mixing carbonaceous material with the oxide and zinc chloride, so that after the mass sets and the chloride has been driven off the portion remaining is immediately ready for smelting by heat or electricity. The porosity of the carbonaceous mixture is an aid in driving off the zinc chloride, as well as in the subsequent reduction. If desired, the carbonaceous mixture could be molded into any convenient form so that after expulsion of the zinc chloride it might be used as electrodes in an electric furnace. Strong zinc chloride liquors can be utilized by precipitating zinc oxide from, say, six-sevenths of their weight and adding the oxide thus produced to the other one-seventh, which would be the requisite quantity to form the oxy-chloride. The density of the oxide formed in the manner described is in some cases twice the original. (1,007,124, Oct. 31, 1911.)

Briquetting Copper Shavings.—According to the invention of Mr. MAX GLASS, of Vienna, Austria-Hungary, metals containing copper, such as shavings and the like, can be briquetted by the use of cupric oxide as a binding agent. If the particles to be briquetted are not already covered with a coating of oxide, they may be heated to 100° C. and thus coated. By treating the mass with ammonia, the cupric oxide forms cuprammonium or ammonical cupric oxide, which diffuses throughout the mass. On pressing and heating, the cuprammonium is decomposed and the cupric oxide deposited uniformly on the surface of the metal where it forms a good cement for the material to be briquetted.

Ammonium carbonate may be used instead of ammonia, a 3 per cent to 6 per cent solution being employed. Quicklime is then added to the treated mass and the briquettes exposed to air after being pressed. The mass heats owing to the slaking of the lime; ammonia is formed which combines with the cupric oxide and diffuses throughout the mass, being subsequently decomposed by the heat of the slaking lime and leaving cupric oxide uniformly distributed over the surface of the particles in a manner favorable to binding the particles together. (1,008,254, Nov. 7, 1911.)

Manganese and Calcium Fluoride.

The process of producing metallic manganese and calcium fluoride by the reduction of manganese fluoride with calcium carbide has been patented by WALTER S. ROCKEY and HILLARY ELDRIDGE, of New York. In a refractory crucible lined with magnesite they place a mixture of manganese fluoride and calcium fluoride, the latter acting as a flux, and fuse the same. Into the fused bath they add a proportionate amount of calcium carbide and continue to heat with exclusion of air. Metallic manganese is reduced and settles to the bottom of the crucible, while the liberated carbon from the carbide floats on top of the molten mass. The reaction is expressed as follows:

$$CaC_2 + MnF_2 = CaF_2 + Mn + 2C.$$

Metallic calcium can be used, but the carbide is preferred on account of the violence of the reaction when calcium is used. (1,007,734, Nov. 7, 1911.)

Roasting Furnace.

Thomas Edwards, of Ballarat, Victoria, Australia, has recently been granted letters patent on an improvement in his roasting furnace, which is of that type in which a series of rotary rabbles is used to stir the ore and advance it from one end of the hearth to the other, owing to the arrangement of the rabbles by which their paths of rotation intersect. The improvement consists in construction which permits the use of the furnace as a combined reverberatory and muffle furnace, or as either reverberatory or muffle furnace only. A cooling hearth also is provided for the discharged roasted ore, on which the same mechanism of intersecting rotary rabbles is used to stir and advance the ore. (1,008,892, Nov. 14, 1911.)

New Electroplating Dynamo.

A new low-voltage dynamo for electroplating and electrolytic work has just been brought out by the Munning-Loeb Company, of Matawan, N. J., under the trade name "Optimus dynamo" (Fig. 1).

Except in the smaller sizes, Optimus dynamos are of the multipolar type in order to secure as good a magnetic distribution and as high an efficency as possible.

As Fig. 2 shows, the Optimus dynamo gives a practically constant voltage from no-load to 25 per cent overload. This remarkable constant line voltage is automatically maintained by means of a special compound winding, without any manual operation of field rheostats. This straight horizontal voltage characteristic makes it possible for the plater to throw different tanks in and out of the circuit at will, without having to worry about the effect on the voltage of the dynamo. There is thus never any danger of burning the edges of the work due to short periods of high voltage.

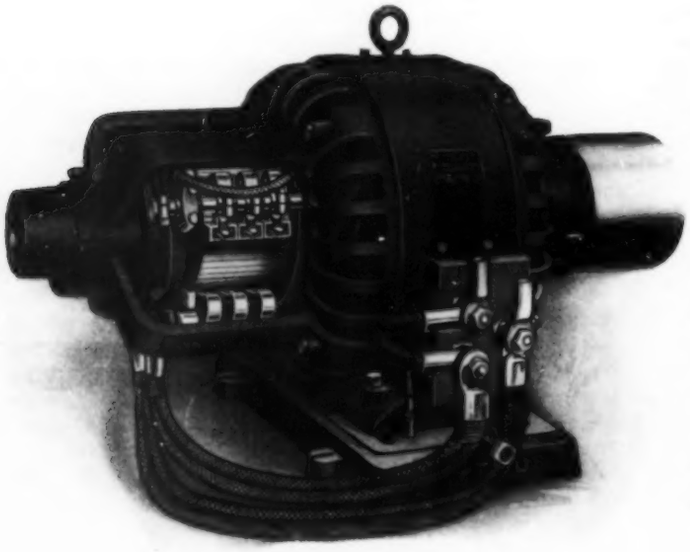


FIG. 1.—ELECTROLYTIC DYNAMO.

Test curves of this generator are also shown in Fig. 2.

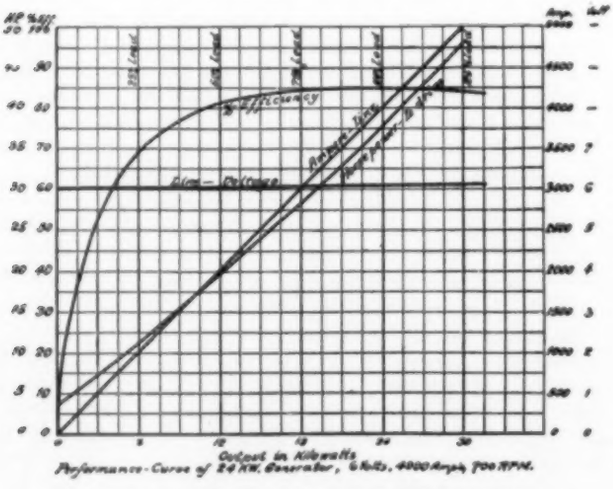


FIG. 2.—TEST CURVES OF ELECTROLYTIC DYNAMO.

The compound winding also maintains a fixed neutral position for the brushes on the commutator so that sparkless commutation is secured without shifting of the brushes with changes in load.

The brushes are made of a special carbon compound and are held against the commutator by means of an adjustable clock spring. A flexible braided copper cable carries the current away from the brushes. These brushes bear radially upon the commutator and it is claimed that they can be used for their entire length and that they will not score or cut the commutator. Furthermore, the fact that the carbon brushes bear radially insures that they will always remain at the neutral point.

The Optimus plating dynamo is built for either two-wire or three-wire circuits in all sizes from 300 watts to 50 kilowatts.

The manufacturers of the Optimus dynamo are also making a special feature of their direct-connected sets, and can supply any size of plating dynamo directly connected to any type of alternating current or direct-current motor, mounted on the same bed plate.

These dynamos are provided with an efficient system of ventilating and the usual guarantee is given that "no part of the machine will have a temperature greater than 45° C. over that of the surrounding atmosphere when the machine is operating continuously at proper speed at full load," although it is claimed that the actual temperature rise given under test is far below the 45° C. quoted in the guarantees.

Alloy Steel Standards.

As a result of the great increase in the use of alloy steels in recent years many chemists whose experience in alloy-steel work has been limited are now being called upon to analyze alloy steels and thus has arisen a decided want for standards which could be relied upon as positively correct.

To meet this requirement, the Scientific Materials Company, of Pittsburgh, Pa., has put on the market four different alloy standards (chrome-vanadium, vanadium, nickel and tungsten steel).

These standards have been checked by seven of the best-known alloy-steel chemists in the Pittsburgh district. The following methods were used.

For chromium: oxidation with potassium permanganate in a sulphuric acid solution, oxidation with potassium chlorate in nitric acid solution, and Blair's method from sodium hydrate precipitate.

For vanadium: Johnson's method by using an internal indicator, Blair's method, and colorimetrically with hydrogen peroxide.

For nickel: potassium cyanide titration and precipitation with dimethylgloxime, and the ether separation method.

For tungsten: by evaporation with hydrochloric and nitric acids.

The analysis of the standards are as follows: The results given are averages of the different determinations and were compiled by Mr. N. B. Hoffman, chief chemist of the Colonial Steel Company.

Chrome-vanadium steel standard (basic open hearth): 0.34 per cent C, 1.06 Cr, 0.16 V.

Vanadium-steel standard (crucible): 0.82 C, 0.228 V.

Nickel-steel standard (basic open hearth): 0.335 C, 3.40 Ni.

Tungsten standard (crucible): 1.25 C, 2.37 W.

Deflocculation.—In a paper read recently in London by Dr. E. G. Acheson before the Society of Chemical Industry the author gave an interesting account of his deflocculated graphite and its uses in oildag and aquadag. Some information given in the conclusion of his address concerning the size of the deflocculated graphite particles which were stated to approach molecular size, was particularly interesting. According to Mr. Jerome Alexander, who examined them by the ultramicroscope, the particles are in rapid motion and of a size estimated as 75 millionths of a millimeter, or 3 millionths of an inch, whereas particles of ordinary finely disintegrated graphite are 1000 times as large.

Uses of Bakelite for Acid-Proof Linings and Other Chemical Purposes.

The General Bakelite Company, 100 William Street, New York City, is putting on the market a great number of varieties of special bakelite raw materials, each one intended for a distinct purpose.

The technique of bakelite varies so much with the condition and with the purposes in view that it is practically impossible to give a general method of working. For instance, the methods used for making molded articles are very different from those which consist in applying liquid bakelite A on the surface of vessels or machinery which have to be protected against the action of chemicals. Then again, the use of bakelite, where high dielectric properties are required, is totally different from those where strength and ease of manipulation are paramount considerations.

For the latter purpose two new varieties of liquid bakelite A have recently been put on the market and a description of these will certainly interest our readers.

These two new varieties (liquid bakelite A10 and liquid bakelite A15) are specially intended for making acid and solvent-resisting linings. They are also very useful as a cement or glue, for instance, in the manufacture of brushes, or for gluing wood, or for applying wood veneer on wood, metal or stone.

The main difference between A10 and A15 is that the former is made with purer materials and has a somewhat greater tensile strength. However, for most purposes, the cheaper variety, A15, is satisfactory. Neither is intended for electrical purposes for which other liquid and solid varieties, bakelite varnish, etc., are specially prepared with a view of high dielectric strength.

The main characteristics of liquid A10 and A15 are the following:

They gelatinize after being heated for a while at a temperature of 75° to 80° C. (167° to 176° F.). If heating at this temperature be continued for several hours a very hard transparent mass of solid bakelite C will be the result. This hardened product will withstand most chemicals and solvents. The process of hardening can be considerably hastened by increasing the temperature after first solidification has set in. The temperature can thus be raised to 90°, 100°, 110° and 120° C. (195°, 212°, 230°, 248° F.) and over, and the hardening process takes relatively little time. Special attention should be paid to the fact that if the temperature be raised too rapidly the mass may become porous.

Liquid A10 or A15 can be spread on a warm piece of wood and heated in a drying stove at temperatures below 100° C. (212° F.) and thus hardened to a very hard brown-red enamel-like coating, which can be sand-papered, polished and finished in the usual ways.

For certain purposes a better coating can be built up by applying a first layer, which after hardening, sand-papering and polishing, is finished by the application of a very thin layer of bakelite lacquer, and then by submitting it to a final baking.

If suitable pigments, for instance, lamp black, vermilion, chrome green, or aniline dye, be added, several suitable colors may be developed at will.

Pieces of wood or metal may be glued together by means of liquid A10 or A15. In this case, the liquids are simply applied as if they were glue. The glued surfaces are kept in contact by means of clamps or other means of establishing gentle pressure, while the objects are heated, either in a baking oven, or any other suitable device.

Here again, the hardening can be carried out at temperatures as low as 75° to 80° C. (167° to 176° F.), and these are the temperatures which should be preferred for objects which can not well stand higher temperatures; but in that case the hardening process is necessarily slow. In all cases where the objects can stand higher temperatures it will be preferable to use the latter and thus shorten the whole process.

Whenever liquid A10 or A15 is used as a glue or as a cementing material it will be found advantageous to incorporate with the same some filling material, for instance, finely ground silex or finely ground sand. If liquid A10 or liquid A15 be mixed with a sufficient amount of short asbestos fiber or clay a paste can be obtained which can be kneaded and worked by hand. Such pasty mixtures can best be made in a sausage machine, meat chopper or other kneading device.

The proportions of filling material and of liquid A vary according to the nature of the filling material and to the consistency of the paste which is desired; this can easily be regulated by increasing the relative amount of either one of the constituents until the mass has the proper physical qualities.

In this way a paste can be made of about the consistency of bread dough, which can easily be applied as a putty or rolled into sheets, and the latter can be applied on metallic surfaces, or can be used to line metallic vessels with a protective coating, which will make them withstand better the action of chemicals or solvents.

The following examples of pasty mixtures may serve as a guide:

1. One part by weight of very short-fiber asbestos and one part of liquid A10 or A15.

2. One part by weight of long-fiber asbestos and one part of liquid A10 or A15.

3. One part by weight of short-fiber Sal Mountain (amphibole) asbestos and one part of liquid A10 or A15.

4. One part by weight of finely ground wood-flour and three parts of liquid A10 or A15.

Pasty material, made as described above, if heated at temperatures of at least 75° to 80° C. (167° to 176° F.), will stiffen by and by to a very hard mass which can be sawed, filed or drilled. The hardness of the mass depends, first of all, on the temperature at which it has been heated.

The temperature of 75° to 80° C. (167° to 176° F.) is considered as minimum temperature, but it is much preferable if heating can be carried at higher temperatures, for instance, 120° C. (248° F.) or over. At the latter temperature the hardening occurs very rapidly and may be finished within one hour, sometimes less. The very best results are obtained by starting at a low temperature and rapidly but gradually increasing this temperature as soon as hardening has set in. Too rapid heating may cause porosity.

The amount and the quality of the filling material has very much to do with the ultimate hardness of the mass; for instance, with long-fiber asbestos the mass will be less brittle than if short-fiber or powdered filling materials be used. The use of relatively long-fiber asbestos gives a very tenacious mass, which can be cut by means of a knife and which withstands shock impacts better. If instead of asbestos clay or slate dust be used the mass will be harder but more liable to break on impact.

In the same way asbestos may be replaced by other fibrous materials, as, for instance, finely ground sawdust or wood-flour. In the latter case the coating will not stand temperatures above those at which wood is decomposed by heat.

For certain purposes where lightness and a certain resiliency are required, finely ground cork, or mixtures of cork and sawdust or wood flour, make an excellent filling material.

Liquid A10 and A15 keep pretty well at ordinary temperatures, although after some months they may thicken somewhat, especially if kept in warm rooms. In the same way a paste made up as described above will keep well for several days and will not acquire its final hardness until it has been heated.

In cases when before hardening by means of heat the paste has become too stiff it can be softened again by addition of fresh quantities of liquid A. Further, liquid A10 or A15 which has become too thick by age can be thinned to the proper con-

sistency by adding some alcohol; but the addition of alcohol and similar solvents will exert a retarding action on the hardening process when heat is applied. So after these solvents have once been added it will be found necessary to heat at decidedly higher temperatures in order to produce maximum hardness.

Liquid A10 and A15 can be diluted with a certain amount of water. For instance, liquid A10 when fresh can take up as much as 1½ times its own volume of water; later on when the product is older and thicker, the amount of water which it can take up becomes considerably less. This singular property of being able to take up water offers a means for diluting the material in case very thin mixtures are wanted.

It should be noted that liquid A15 takes up considerably less water than A10, so that in this case water has to be added with more precaution and more sparingly. At any rate, whenever it be found desirable to add water, the addition of the latter ought to be done carefully while stirring, and no more water should be added than what will give a transparent solution. The addition of more water will cause a white precipitate, and a separation of the mass into a thick, viscous, white, grayish substance.

Liquid A10 or A15, exposed to the air, develops a dark red-brown color, which is very characteristic.

As already stated, the pasty mixture of liquid A10 and A15 can be used as a cement or as a packing material. If it be applied to a hot steam pipe it will soon harden to the point where it only can be detached by means of a chisel.

It also can be applied so as to protect from corrosion metallic vessels or metallic machinery submitted to the action of chemicals. The application is easily made by rolling the paste out in sheets of about a quarter of an inch thick or more and applying these sheets to the metallic surfaces which have to be protected, then baking the whole in an oven with the precautions as described above, until the mass has been transformed into the final condition of maximum hardness.

For such purposes it is always better to apply the material in not too thin layers because it is very difficult to spread a thin film of any material so uniformly that there should not exist some unprotected places where corrosion can occur.

For certain chemical uses like digesters the possibility of chemical corrosion is augmented by mechanical abrasion due to the friction of moving liquids or solids. For such purposes there exists a simple and efficient way for lining the vessel with acid-proof bricks held together with bakelite cement.

This is accomplished in a somewhat similar way as porcelain-lined ball mills are made, with the difference that the walls of the vessel ought first to be lined with a thin sheet of asbestos paper, impregnated with liquid A; on this the brick lining is inserted and fastened with a mixture of liquid A and some inert filling materials like clay baryta, or powdered silex, or asbestos. This liquid mixture is easily applied and then a suitable heater, for instance a gas stove, is introduced in the vessel so as to raise the temperature of the lining to at least 80° C. (176° F.), or higher, until the polymerization into bakelite C has taken place.

Whenever asbestos is used as a filling material it should not be forgotten that Canadian asbestos is easily attacked by acids. Amphibole asbestos is considerably more resistant to acids; unfortunately it lacks strength and fibrous qualities. Green Cape asbestos resists acids sufficiently well and has a good strong fiber, but it is more expensive.

Coal Age is the title of a new weekly devoted to coal mining and coke manufacture and published by the Hill Publishing Co. The first issue appeared on October 14. The issues so far received show that the field is broadly covered both as to the technical problems and commercial news. An interesting new feature is the sociological department. We cordially wish our contemporary a long and successful career.

The Electric Furnace for Small Steel Castings.

Neglecting malleable, semi-steel, and other kinds of castings masquerading under the good name of steel castings, there have been in the United States until recently only three processes for the manufacture of steel castings.

All specifications to date call for steel castings to be manufactured by one of the following processes: crucible process; Tropenas or similar processes; open-hearth process.

It is quite usual to divide commercial steel castings into three grades, viz.: soft, medium and hard. The chemical characteristics corresponding to these three grand divisions are in general as follows:

	Soft.	Medium.	Hard.
Carbon	0.10-0.20	0.20-0.35	0.35-0.50
Manganese	0.20-0.85	0.20-0.85	0.20-0.85
Phosphorus, not over	0.05	0.05	0.05
Sulphur, not over	0.05	0.05	0.05
Silicon, not over	0.40	0.40	0.40

The corresponding physical characteristics when the steels are properly annealed are as follows:

	Soft.	Medium.	Hard.
Tensile strength, minimum, in lbs. per sq. in.	55,000	65,000	80,000
Yield point, minimum, in lbs. per sq. in.	27,000	31,000	38,000
Elongation, per cent	22	18	15
Reduction, per cent	30	25	20
Bending test	120°	90°	Slight

As a general proposition the essential characteristics of the majority of steel castings are as follows:

First.—Castings well cleaned, secured by sand blasting.

Second.—Solid castings, secured by proper gates and heads.

Third.—High yield point, approximating the ultimate strength, secured by soft or medium grade of castings.

Fourth.—A good bending test, secured by soft or medium castings.

Fifth.—Freedom from brittleness when cold, secured by low phosphorus.

Characteristics 1 and 2 are within the control of every foundry. Moreover, defects in these qualities can usually be detected by the consumer through surface inspection.

Qualities 3, 4 and 5 can only be determined by the consumer through laboratory tests, and, in the absence of such tests, by care in selecting the manufacturer and the process he uses.

Crucible-Steel Castings.

The manufacture of crucible-steel castings as practised in this country is purely a melting process, requiring little or no technical knowledge, as is evidenced by the rapid, almost mushroom growth in the number of plants throughout the country and clearly shown in the records of the advertising pages of technical journals.

Crucible-steel castings are, as a rule, made from punchings purchased in the open market. Unless careful analysis is made of each and every car bought, most varying results in the castings are apt to follow. In this process there is little or no change in the phosphorus and sulphur during the melting of the punchings. The percentages of silicon and manganese are more or less under the control of the melter. The increase in the percentages of carbon is the most serious drawback to this process; in general it increases about two-tenths of 1 per cent.

The average good crucible steel casting will analyze:

Silicon	0.25 to 0.50
Manganese	0.40 to 0.80
Phosphorus	0.025 to 0.040
Sulphur	0.030 to 0.040
Carbon	0.35 to 0.50

The percentage of carbon in the punchings *may* remain constant during the melting, if the raw material is melted most carefully in a clay or clay-lined pot. This is rarely or never done in commercial practice, owing to the added initial cost of the pots and the fact that they last for fewer heats than the regular graphite pots.

If punchings of the same analysis are melted in graphite pots, the resulting steel will vary from pot to pot, and will

also vary when melted in the same pot, depending upon how often the pot is used.

The change that takes place is well illustrated by the following actual test, which was an effort to make low-carbon steel castings, the material charged in the pot being selected muck bar:

	Analysis of Muck Bar.		Analysis of Casting.
Silicon	0.21	Silicon	0.17
Manganese	0.00	Manganese	0.39
Phosphorus	0.264	Phosphorus	0.204
Sulphur	0.017	Sulphur	0.015
Carbon	0.09	Carbon	0.29

From what has preceded it is to be noted that commercial crucible steel castings (owing to high carbon content) come under the grade known as hard. They are usually difficult to machine and will stand little or no bending test. Due to the fact that the crucibles hold only 80 to 100 lb. of metal, more or less variation in the chemical and physical qualities is bound to result in a given lot of castings. While one casting may possess the desired qualities, a similar casting poured from the succeeding pot may be far below the standard.

Many consumers of small steel castings have purchased them from the foundries using the crucible process, under the false impression that they compared in quality with "crucible steel," otherwise known as "tool steel."

Tropenas Steel Castings.

The Tropenas, like the crucible process, is a melting, rather than a refining, method for making steel.

Tropenas steel castings are made by first melting the material in a cupola and then changing it to steel by means of a converter. A typical charge for the cupola would consist of about 63 per cent of pig iron, 20 per cent of steel or wrought scrap, and about 17 per cent return scrap, that is to say, sink heads and gates from steel castings.

It is to be noted that the resulting product of the cupola comes more or less under the head of what is known to the trade as semi-steel. This semi-steel is tapped from the cupola into the converter and by means of a blast of air the carbon, silicon, and manganese are burned out. Ferromanganese, ferrosilicon or silicon spiegel are then added to give the desired amounts of carbon, manganese, and silicon.

The Tropenas process is essentially the old well-known Bessemer process under a new name and on a smaller scale. Its main claim to superiority over the latter being that the air for combustion is not blown through the metal but on the surface of it. Like the Bessemer, it is also inferior to the open-hearth process.

Without the most careful selection of coke for melting in the cupola, high-phosphorus and high-sulphur steel will result. Great care must be exercised in selecting and analyzing the pig iron used, as the phosphorus and sulphur therein are not removed. The failure or success of this method of making steel castings is largely dependent upon the skill of the particular melter in charge.

Steel made by this process has the valuable characteristic of being extremely fluid owing to its high temperature. It is not, however, as well adapted to the making of small steel castings as the crucible process, because the metal must be taken from the converter and poured into the mould from ladles which are at a much lower temperature than the converter, whereas in the crucible process the steel is poured from the crucible in which it was melted.

The Tropenas process, as generally practised, overcomes the defects of the crucible process, in that the carbon is more or less under control and lower carbon steel can be made, but great and wide fluctuations may occur from day to day in the different plants in the percentages of phosphorus and sulphur.

Below are given the analyses of castings from two different plants:

Silicon.	Manganese.	Phosphorus.	Sulphur.	Carbon.
0.41	0.29	0.052	0.050	0.26
0.24	0.71	0.100	0.043	0.32

As far as a commercial proposition is concerned, and as far as the day in and day out practice of making *good* steel castings is concerned, neither the crucible nor the Tropenas process seems to exactly fill the bill. The majority of the purchasers of steel castings buy without the rigid requirement of specifications, because to meet the specifications the manufacturer generally exacts an increased price, and therefore it is this day-in and day-out practice that the average customer is interested in.

To the purchaser who requires soft or medium castings the crucible process holds out little hope, and in the Tropenas process the case is little, if any, more encouraging because of the phosphorus and sulphur.

In the former, the manufacturer hopes as to his carbon and softness of his castings, while in the latter he does the best he can with the materials on hand and trusts his castings will not be too cold short.

The quality of steel castings made by the open-hearth furnace is far superior to those made by either of the processes mentioned, but, owing to the difficulty in handling the product, it is not generally used in the making of small steel castings in this country.

Electric Furnace Castings.

The needs for and the increase in the number of electric furnaces are closely related to the introduction of the automobile and the requirements of the builders thereof.

The electric furnace, like the automobile, came in with the century. In 1904 there were only four electric furnaces in Europe, whereas there are now over one hundred.

Unlike the automobile in this particular, however, while America produces the greatest number, it is far behind Europe in the adoption of the electric furnace. Foreign automobiles are held at a premium in the United States and many manufacturers here go to Europe for materials which, they say, cannot be procured in similar quality in this country.

An examination into Europe's metallurgical methods bears evidence to the fact that many crucible steel plants have discarded and abandoned their crucible furnaces and installed electric furnaces. In the United States electric furnaces have been installed for the making of tool steel or refining steel made by the Bessemer or even the open-hearth process.

While electric furnaces require for their successful operation a chemical and physical laboratory and the best of technical knowledge, they do make good steel starting with either hot or cold raw material of any composition, pig iron, steel scrap, or even from iron ore. The secret of success is in the procuring of a high temperature, the absolute control of the percentages of the elements which go to make up good steel, and the fact that the bath of molten steel can rest a long time underneath a neutral slag, permitting the removal of any gases or included slag, and the proper co-ordination of these various factors.

Electric furnace steel can be made with any carbon content, phosphorus can be reduced when desired to between 0.01 and 0.02 per cent and sulphur can be kept within the same limits. In fact, as stated, all the elements are within absolute control, as the metal need not be tapped from the furnace until its chemical and physical character meet the wishes of the melter, and in turn correspond exactly with the specifications of the consumer.

These valuable features are found in no other known metallurgical process for iron or steel, nor are they so easily attainable, nor to such a degree, by other means as they are in the electric furnace.

Self-Cleaning Still.

In distilling apparatus for water the impurities from the water usually concentrate and accumulate in the bottom portion of the retort of the still. At regular intervals they must be removed either by hand or are drawn off through a valve-controlled opening in the bottom of the retort. In both cases the continuity of operations is impaired.

This disadvantage is overcome in an automatic self-cleaning still recently devised by Mr. W. H. McCune, chief chemist of the American Sheet and Tin Plate Co., of Vandergrift, Pa. The details given in the following article are taken from Mr. McCune's U. S. patent 1,010,508, of December 5, 1911. The essential feature is the provision of automatic means for automatically flushing the retort of the still at predetermined interval, for the removal of the collected solids. The length of the intervals in which this happens can easily be adjusted as will be seen.

Fig. 1 is a vertical cross-section of the distilling apparatus in which 2 is the copper retort with tinned inner surface

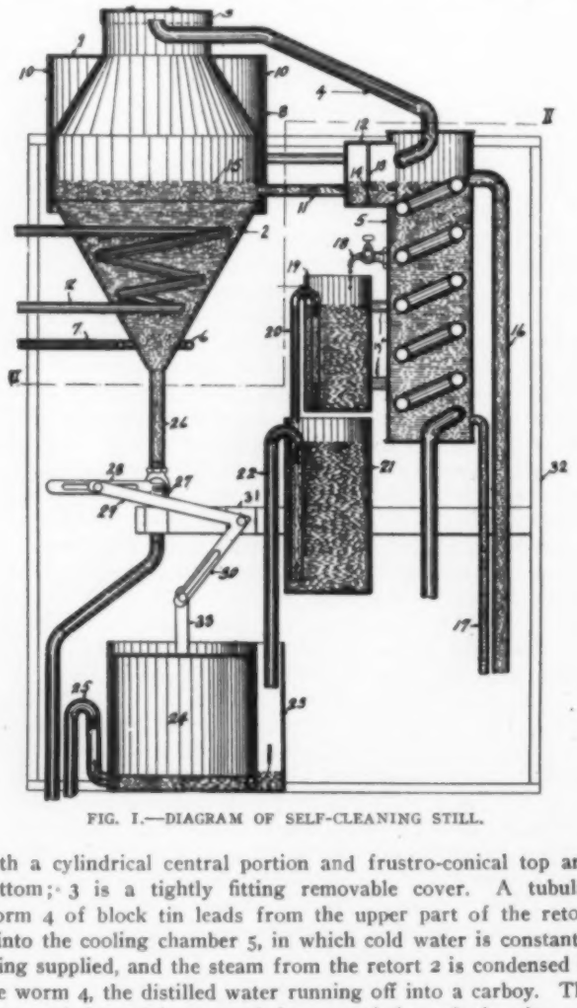


FIG. 1.—DIAGRAM OF SELF-CLEANING STILL.

with a cylindrical central portion and frustro-conical top and bottom; 3 is a tightly fitting removable cover. A tubular worm 4 of block tin leads from the upper part of the retort 2 into the cooling chamber 5, in which cold water is constantly being supplied, and the steam from the retort 2 is condensed in the worm 4, the distilled water running off into a carboy. The retort 2 is heated by steam being passed through the pipe coil 2′ or by an annular gas burner 6. The products of combustion are carried off through the hood 8.

The retort 2 and the cooling chamber 5 are connected together by means of the horizontal overflow pipe 11 and box 12 which contains a vertical partition 13 with a series of small holes 14 of such size as to permit water to flow from the cooling tank 5 over the retort at about the rate at which the water is vaporized in the retort. The holes 14 are, therefore, at a level slightly below the level 15 at which the water is maintained within the retort 2.

The cooling chamber 5 is further provided with a water overflow pipe 16 at the top and a water inlet pipe 17 at the bottom.

From the stop cock 18 water is permitted to drop from the cooling chamber at a prearranged rate (about which more will be said below) into a tank 19 provided with a syphon 20.

Beneath the tank 19 is a similar tank 21 of three times the capacity of tank 19, also provided with a syphon 22.

The syphon 22 discharges into the float tank 23, which is provided with a float 24 with feet on its bottom to keep the bottom of the float out of contact with the bottom of tank 23. A U-tube 25 forming a syphon is connected to the bottom of the float tank 23 so as to practically drain this tank when the syphon 25 is started in operation.

The lower, smaller end of the conical bottom on the retort 2 is provided with a drain pipe 26 and within this drain pipe is a stop cock 27 which is normally in closed position during the operation of the apparatus. The stop cock 27 is provided

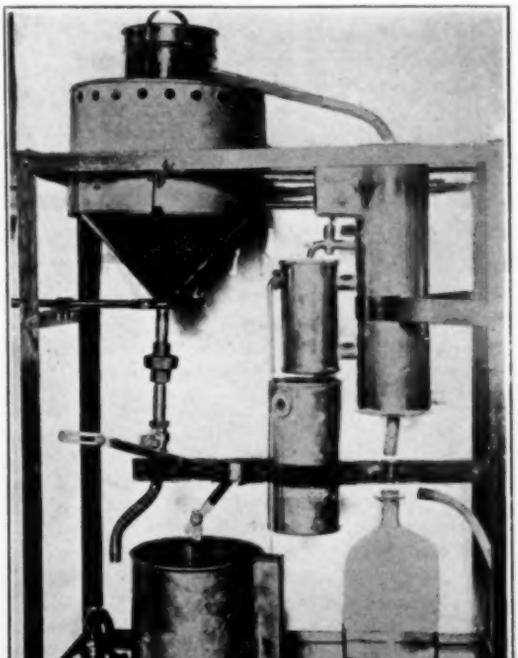


FIG. 2.—OUTSIDE VIEW OF STILL.

with an operating lever arm 28 connected to one end 29 of the bell crank formed by the levers 29 and 30. The bell crank is pivotally connected to a support 31 which is secured to the drain pipe and the framework 32 supporting the apparatus. The outer end of the lever arm 30 of the bell crank is connected to the rod 33 on the top of the float 24 so as to open and close the stop cock 27 with the rising and falling of the float in the tank 23.

So much about the construction of the apparatus. Its operation will now be easily understood. It is as follows:

Water is supplied through the pipe 17 and cooling chamber 5 to the box 12 and through this box and the pipe 11 to the retort 2, which is being heated. When the water becomes heated to the boiling point the steam passes upwardly in the retort and is led downwardly through the worm or condensing coil 4, and after the steam is condensed the resulting distilled water is led outwardly through the condensing cham-

ber 5 through the lower end of the condensing coil and is delivered into a carboy or other receptacle in readiness for use.

A supply of fresh water is constantly provided in the retort 2 through the pipe 11 and box 12 which connect the retort with the condensing chamber 25, and the condensing chamber or tank 5 is being constantly supplied with fresh water through the pipes 17, the water being constantly maintained at the desired level in the apparatus by means of the overflow pipe 16.

The stop cock 18 is opened so as to drip water slowly into the tank 19. When the tank becomes filled with water the syphon 20 operates and discharges the water in the tank 19 into the lower tank 21. Since the lower larger tank 21 has a capacity of about three times that of the upper tank 19, the tank 19 will be emptied three times before the larger tank 21 is filled sufficiently to start the syphon 22 to operate and deliver the contents of this tank 21 into the float chamber 23.

The function of the syphon will be more readily understood by first assuming a construction in which the vessel 19 and syphon 20 are absent. In such case, where water is supplied drop by drop to the vessel 21, when the water reaches the level of the top of the lower curve of the bend in the syphon 22, it will simply flow over in a thin film, no syphonic action, properly speaking, taking place. But when the contents of the tank 19 are quickly supplied to the tank 21 and the tank 21 is sufficiently full, the syphon 22 is filled to the top of its bend and a true syphonic action takes place whereby the float 24 is quickly lifted to be as quickly allowed to sink, the chamber 23 being rapidly emptied by the relatively large syphon 25.

When the water in the tank is delivered into the float tank 23, the float 24 is caused to rise in the tank 23, and through the bell crank connection formed by the rigidly connected levers 29 and 30 moves the lever 28 on the closed stop cock 27 from the horizontal position shown in Fig. 1, so as to extend vertically. When in this position the cock 27 is open and water in the bottom portion of the retort 2 will be drained from the retort. When this occurs the solid matter collected in the bottom of the retort will be carried out of the retort by the water.

When the float tank 23 has been filled with sufficient water the syphon formed by the U-pipe 25 will be started to operate and the water drained from the float tank 23.

As the level of the water in the float tank is lowered the float 24 descends in this tank and, through the connecting bell crank mechanism, closes the stop cock 27.

The frequency with which the retort 2 is drained is regulated by varying the volume of water discharged from the chamber into the syphon tank 19 by the drip cock 18 so that the number of times the retort 2 is flushed in a given time can be readily changed.

When a quantity of water has been removed from the retort 2 by flushing the retort through the drain pipe 26 the water again regains the desired level in the retort 2 in a short time (ordinarily about two minutes with the apparatus illustrated) and a continuous operation of the apparatus is practically effected.

Fig. 2 shows a still of this type which is now in regular operation. It is heated by gas alone from the outside. The efficiency of the still, however, is greater when the water is heated from the inside by means of a steam coil or by an electric heating coil.

Faraday Society.—At a meeting held in London on December 6th the following papers were presented: Dr. F. J. Brislee on the density and coefficient of lineal expansion of aluminium; Dr. V. H. Valez, on the solution volumes of nitric acid; Dr. J. N. Pring and Mr. J. R. Curzon on the influence of the physical condition of metals on cathodic overvoltage; Mr. W. R. Cooper on the Benkoe primary battery; Prof. H. Marshall on thermostats, Mr. W. R. Bousfield on thermo-regulators, and Dr. A. Cumming on thermostats, with exhibitions of these instruments by Dr. Lowry.

Metallic Roofing.*

BY JAMES A. AUPPERLE.

Experience teaches that there are ten fundamental principles which every engineer considers most carefully before drafting specifications for a roof. The ideal roof must conform to all the following requirements:

It must be (1) weatherproof, (2) fireproof. It must have (3) minimum weight and (4) maximum strength. There must be (5) provision for expansion and contraction. (6) The first cost must be low. Other requirements are (7) simplicity and low cost of application, (8) low insurance rates, (9) minimum cost of maintenance, and (10) long life and durability.

There are many excellent grades of roofing which fulfill some of these requirements perfectly, but a systematic, thorough comparison of all grades proves most conclusively that metallic roofing, as manufactured from pure iron, conforms to all the requirements and meets every condition better than any other kind of roofing manufactured to-day.

The following facts will demonstrate how uniformly the pure iron roofing meets each important requirement.

Weatherproof.

The many different styles of metallic roofing permit its application to all kinds, shapes and forms of roof construction, providing the necessary and satisfactory grade for each particular roof and the variation in pitch.

The joints are soldered, double-seamed or lapped so snugly that they are perfectly watertight.

Driven snow cannot penetrate these joints and they are impervious to capillarity which, in freezing weather, plays such havoc with slate and tile roofs.

Because of the large sections in metallic roofing there are less seams and joints exposed than in practically any other grade. There is absolutely no absorption of moisture and its destructive consequences. Therefore a metallic roof is perfectly weatherproof when properly applied.

Fireproof.

From a fireproof standpoint metallic roofing is the peer of all roofing. It is not only absolute protection against fire from without but holds and smothers the fire within, thus preventing the flames from spreading to adjacent buildings.

The lightness of this roofing obviates the danger of a heavy roof crashing through the building before the supporting timbers are half consumed.

It may be coincidental, but an investigation to learn the effect of lightning upon metallic roofing resulted in the discovery that of the many consulted there was no record of a metallic roof having been struck by lightning.

Page 702 of Foster's "Electrical Engineers' Pocketbook" contains the following statement: "A sheet of copper, a sheet of iron, a tin roof, if without breaks and fully connected by well-soldered joints, can be utilized to advantage as lightning conductors. Many architects take this into consideration, making the eaves trough and downspouting form a perfect connection to the earth."

Weight.

Comparing the weight of metallic roofing with all other roofing proves that it is as light as the very lightest forms and very much lighter than the more substantial forms of roofing such as slate, tile and concrete.

As an example, No. 20 gauge corrugated roofing, galvanized, weighs 1.78 lb. per square foot. A reinforced tile roofing weighs 13½ lb. per square foot.

This introduces one of the strongest features in favor of metallic roofing, viz., the great saving in the cost of the superstructure.

All corrugated roofing can be directly applied to the purlins.

*A paper read before the Engineers' Society of Western Pennsylvania.

V-crimp, pressed standing seam and roll roofing require a very light sheathing, while in all cases a very light and less costly superstructure is perfectly adequate.

Strength.

In this respect metallic roofing again excels, presenting the very strongest form of roofing obtainable. This feature is of the utmost importance when wind pressure, snowloads and walking upon a roof are taken into consideration.

In practically all forms of metallic roofing the seams and joints are interlapped over the entire surface and the straps and cleats are in no wise exposed.

This construction has two important advantages. Firstly, the fastenings are not subjected to deterioration, and no part of the roof is dependent upon one, or a few, fastenings which may give way and cause breakage and leaks as in tile and slate, but all the fastenings act as a unit in keeping it in place. Secondly, it is absolutely impossible for the wind to get under it.

Very few roofs are constructed over which it is not necessary for someone to walk at one time or another, and no roof offers better resistance to this kind of abuse than does the metallic roof.

Corrugated roofing, because of the known tensile strength of iron, and the strength imparted by the corrugations, is not only self-supporting but is undoubtedly the very best form of roofing for large structures, mills and factories of all kinds. It is the strongest roof made.

Expansion and Contraction.

The extremes of heat and cold are amply provided for in metallic roofing. The corrugations, seams and joints are all constructed to meet this particular requirement. It is, therefore, impossible for it to crack or pull apart, but, on the other hand, it gives and returns without injury to the efficiency of the roof or tearing away from the fastenings.

First Cost and Insurance.

According to the demands and requirements of the trade metal roofing gives as low, if not a lower, first cost due to the low price of the material itself. The ease and simplicity of application emphasize this feature and it possesses the minimum possibility of poor workmanship.

The highest tribute to metallic roofing is presented by the insurance writers. In this respect it represents a great saving because it receives the lowest rates. The fact that the National Fire Protective Association places metal roofing first proves its great superiority from a fireproof standpoint.

Cost and Maintenance.

The first cost can only be correctly computed by anticipating the cost of maintenance. Those roofs which often appear cheapest are, in the long run, most costly. Because of the features found in metallic roofing, previously mentioned in this paper, the upkeep of this form of roofing compares most favorably with every other form. As has been stated, leaks can be easily found, permanently repaired, and at a minimum expense. It is waterproof, vermin-proof and does not become watersoaked, thus preventing an increased load on the superstructure. It will stand more abuse, greater extremes of temperature. It permits of remodeling, additions, and even fire does not destroy its utility.

Life and Durability.

The only just criticism offered against a metallic roof is its lack of durability. No defence can hide the fact that many metallic roofs made of steel have proved a dismal failure because of their rapid deterioration.

However, a careful review of the history of metallic roofing develops the fact that the roofs made fifty and sixty years ago did not possess this fault. On the contrary, many of these roofs are in service to-day and are still in good condition.

Further investigation into the quality of metal used proves that they were made from iron.

Some attempt has been made to explain this by calling attention to the coating applied, and claiming the life of a roof depends upon the coating and not upon the base metal. If this is true, why do galvanized, painted and tin roofs of to-day deteriorate so quickly, while these old iron roofs, sometimes on the same building, continue to withstand the ravages of time? One might surmise that the coating of to-day is inferior, but old, experienced coaters say not.

There is just one sane conclusion to be reached, and it is this, that the durability and long life of metallic roofing depend essentially upon its being made of a base metal free from those ingredients which accelerate corrosion. The electrolytic theory of corrosion is generally accepted and the impurities which hasten decay must be eliminated if long life and durability are to be attained. Having obtained the best possible base metal it is necessary that it be coated with the best protective coating obtainable.

The question has been asked, "If the base metal is so good, why coat it at all?" But the same question might be applied to other material by asking "Why coat a cedar fence-post, when placing it in the ground?" The answer is obvious. But if you paint a poplar post it would hardly follow that this post will last as long as a cedar post receiving the same protection. Nor does it follow that coated steel, full of impurities, the natural accelerators of corrosion, will have the life of a coated pure iron, made to resist corrosion to the greatest possible degree.

Old-fashioned pure iron roofs were uniformly reliable. This fact has brought forth a modern pure iron, purer than the old-fashioned iron, and, although it has been in service only approximately five years, the intense and universal satisfaction it is giving in places where steel failed under the same conditions is quickly overcoming the one criticism of metallic roofing and eliminating entirely the only unsatisfactory feature.

American ingot iron[1] can be made in larger units and consequently at a lower cost than the well-made old-fashioned iron, and because of this fact it is within the means of all at a price very little higher than steel.

A brief summary why this iron possesses the required durability is as follows:

The chain from the Newburyport Bridge across the Merrimac River when examined under the microscope shows the well-known ferrite structure almost free from slag. The oxygen content is but slightly higher than ingot iron. The high purity and comparative freedom from slag and the 100 years' service of this chain prove that the purer the metal the better it will resist corrosion.

According to available records the Newburyport Bridge had not been painted for seventy years and even if it had been frequently painted we should expect the paint to wear very rapidly where the links joined, due to the expansion and contraction as well as the vibration from traffic.

The density or specfic gravity of iron has considerable influence upon the rate of corrosion; the more porous the material the more rapidly will corrosion proceed, and the deeper and more destructive will it be. As first discovered by Kalischer (A. Sang, "Corrosion of Iron and Steel," p. 43) metals conduct electricity better when their structure is crystalline. Increased conductivity implies better resistance to corrosion. American ingot iron has 50 per cent higher conductivity than average steel.

The consumer is often unable to judge which grade of material will give him the best service. There are many different grades of iron and steel on the market such as puddled iron, piled iron, ingot iron, acid and basic Bessemer, and acid and basic open hearth, and in addition the manufacture of the same kind of steel varies at different furnaces.

[1]See, for instance, this journal, vol. VIII, pp. 61 and 98.

If the consumer gives an iron of known purity a practical test and in future years desires to procure the same material he will know exactly the kind of material that has given him satisfaction.

Protective Coatings.

All iron roofing should be galvanized or painted. It has been found that ingot iron dissolves in molten spelter about one third as rapidly as steel, consequently the zinc does not become contaminated with as much dissolved iron.

That purity of spelter is of great importance from a corrosion standpoint has been proven, and we have made careful tests on the spelter coating of American ingot iron and have found that it resists the action of sal ammoniac, ammonia, dilute hydrochloric acid and caustic potash or soda better than the coating on steel. The most important point, however, is that spelter on ingot iron is from 200 per cent to 400 per cent more resistant to the action of boiling water than the spelter on steel.

In the process of manufacture of ingot iron highly oxidizing influences are required in order to eliminate the impurities which exert a marked influence upon the physical properties of iron as well as upon its rust-resisting properties. The influence of constituents usually unlooked for in ordinary chemical analysis, such as oxygen, hydrogen and nitrogen, is not fully understood. We should expect, however, that iron or steel high in oxygen will corrode rapidly due to the depolarizing effects of oxygen.

The Effect of Manganese.

Numerous investigators have studied the effect of manganese upon the corrosion of steel. Siemens (*Inst. Journal*, 1878, Vol. I., p. 44) believes that the presence of manganese in steel increases the readiness with which it corrodes. Snelus (*Inst. Journal*, 1881, Vol. I., p. 66) ascribes the pitting in steel to the irregular distribution of manganese in the metal. Turner ("The Metallurgy of Iron, p. 421) has observed that certain samples of manganese-steel rust more readily than any other variety in the collection of specimens at the University of Birmingham, while it is well known that rich ferromanganese when exposed to moist air oxidizes with extreme rapidity. Dudley (Sang, "Corrosion of Iron and Steel," p. 84) discovered some years ago that segregated manganese formed centers of corrosion and it is a generally accepted fact that steels high in manganese are peculiarly liable to oxidation.

Cushman ("Corrosion and Preservation of Iron and Steel," p. 78) has carried on some careful investigations on the effect of manganese in steel, from which we note the following: "From the standpoint of the electrolytic theory there are several reasons why the presence of manganese in steel should invite corrosion. Manganese decreases the electrical conductivity of iron, and as the percentage of manganese, starting from zero, rises the electrical resistance increases up to a certain specific maximum. It will be seen that if the presence of manganese in iron raises the electrical resistance any variation in the distribution of the manganese means that there will not be a constant electrical conductivity throughout its mass or on any given surface. There is abundant evidence to show that manganese associates itself, to a considerable extent, with sulphur when both these impurities are present in steel. That manganese sulphide shows a difference of electrical potential against iron is also well known."

These theoretical reasons, which indicate that manganese should stimulate corrosion of steel, provided it is not homogeneously distributed in the iron, appear to be justified by a number of reliable observers, among whom may be mentioned Dudley, Drown, Abel and Reynolds.

Huntley (*Journal Society of Chem. Ind.*, Vol. 28, p. 339.) deals with the corrosion of boiler steel and gives details of a case of pitting in a boiler: "Each pit was found to be the center of a blister, and the blister contained a slightly acid solution of ferrous sulphate, while the boiler water was alkaline with caustic soda. The solid matter in the envelope of the blister, which consisted of a mixture of iron oxides, acted as a semipermeable membrane, keeping apart the ferrous sulphate within and the caustic soda outside the blister. The theory advanced is that the particles of manganese sulphide segregated in the steel were oxidized by the oxygen dissolved in the boiler water to sulphuric acid and an oxide of manganese, the acid then acting locally on the surface of the boiler plate in the vicinity of the particles of sulphide of manganese. Manganese also increases the occlusion of gases in steel. It is also known to fix oxygen on certain compounds, and that even the solid salts at suitable temperatures hasten the oxidation of many substances. The metal itself will precipitate iron from its solutions and it is reasonable to infer that with iron going into solution in the presence of oxygen if there is any manganese present it will aid its precipitation as an oxide."

Stoughton (*Engineering Magazine*, July, 1911) on the corrosion of iron and steel has the following to say: "In short, manganese is only one, although perhaps one of the worst, of the evils we have to contend with. We should expect to find that the most resistant metal would be the one which was the most homogeneous in composition and because the less the amount of impurity the less the opportunity for an uneven distribution of it, we should expect that the purer the metal the more resistant it will prove to be. Both of these theories are borne out by the facts as we know them, and it has been found good practice to reduce all impurities—not only manganese, but sulphur, phosphorus, silicon, carbon and also gases which may occur with the metal, occluded in it, or separated in the form of blow-holes. It will, therefore, be readily understood that any irregularity in the composition of the metal which produces a difference in potential between two parts, no matter how close together, will increase the rate of corrosion, and this no doubt is one of the important reasons why metal charged with blow-holes or occluded gases or in which the impurities are segregated has rusted faster than better made material.

American ingot iron, especially if the manufacturers of it have succeeded in also avoiding blow-holes, oxides and slag, is a very advantageous feature and should recommend it to users on theoretical grounds at least."

Influence of Copper.

The addition of copper to ordinary steel has the property of retarding the solution of the steel in sulphuric acid. Copper is insoluble in dilute sulphuric acid and exerts a restraining influence upon the solubility in sulphuric acid of either iron or steel.

Comparisons have been made between copper and non-copper steels by Stead and Wigham (Iron and Steel Inst., 1901, Vol. 2, p. 135), who made tests on copper steel by placing samples in the Middlesborough River. While the results are somewhat confusing the steel containing the highest percentage of manganese and sulphur showed the greatest loss but this sample incidentally contained the lowest percentage of copper, while another steel, containing a trace of copper, showed less loss under the same conditions than the steel containing almost 1 per cent of copper.

Another fact very noticeable in this test is that the steel containing the lowest total impurities (excepting copper) showed the lowest loss.

Howe ("Metallurgy of Steel," p. 83) states that some Bessemer steel contains a high percentage of copper, while Campbell ("Metallurgy of Iron and Steel," p. 358) has shown that most of the Bessemer steels recorded in his book contained from 0.30 to 0.50 per cent copper.

The careful experiments conducted at Atlantic City show that Bessemer steel sheets corroded very quickly and invariably with a peculiar form of corrosion, to which Mallet (Turner, "Metallurgy of Iron," p. 418) gave the name of "tubercular corrosion," which he states is due to irregularity in the composition of the original metal, "The result is the formation of

mounds of rust, with pitting of the metal underneath. This form of corrosion is not infrequently met with in tubes, boiler and ship plates and other iron work and is usually very rapid in its destructive action."

If Bessemer steel invariably contains copper, as has been noted, and if it corrodes in this rapid tubercular manner the presence of copper did not retard natural corrosion while it may have retarded acid corrosion.

I have endeavored to show the harmful effect of impurities from a corrosion standpoint and I have shown that the presence of slag is not an indication of resistance to corrosion, as the Newburyport Bridge chain, after 100 years' service, is found to be almost free from slag, and of almost the same composition as ingot iron made in open-hearth furnaces.

I have also shown the influence of copper, manganese and gases and will now compare the analyses of ingot iron and the chain from the Newburyport Bridge.

	Newburyport Chain	American Ingot Iron
Carbon	.0500	.0150
Manganese	.0050	.0100
Silicon	.0190	.0040
Sulphur	.0070	.0200
Phosphorus	.0320	.0030
Copper	.0070	.0500
Oxygen	.0580	.0270
Nitrogen	.0033	.0035
Hydrogen	.0005	.0006

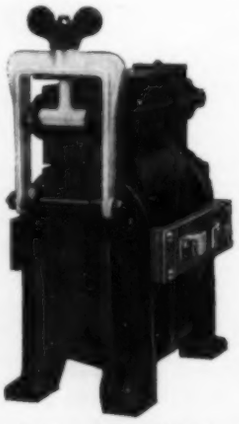
Having thus shown the high purity of both metals adds more evidence to the correctness of the electrolytic theory of corrosion and shows the necessity of removing those elements from iron which differ from it electrochemically, which is being done in the manufacture of American ingot iron.

American Rolling Mill Company,
Middetown, Ohio.

Flexible Step-Up Transformers.

The use of the small high-potential transformer for commercial purposes is increasing continuously. All up-to-date electrical factories, whether they are small or large, have one or more step-up transformers for testing their products.

A high-potential transformer that is in use continuously must be constructed to stand hard usage and so designed that there is little or no chance of burning it out.



STEP-UP TRANSFORMER

The adjoining illustration represents a type of transformer built by the Thordarson Electric Mfg. Company, 222 South Jefferson Street, Chicago, Ill., that answers all the requirements for commercial use. They are small and compact and all parts are exposed. They are designed to be run continuously, if necessary. They are constructed with a magnetic shunt and equipped with a regulating device to give instant adjustment.

No impedance coil or other resistance is necessary. These transformers are designed for connection to any alternating-current circuit. They are self-regulating as far as the current consumption is concerned.

When more power or capacity is required, all that is necessary is to simply turn the thumb screw, there being no necessity to remove or loosen bolts or nuts, etc.

These transformers are finding especial favor for generating ozone, for testing insulation, for electrostatic separation, wireless telephony and telegraphy, etc.

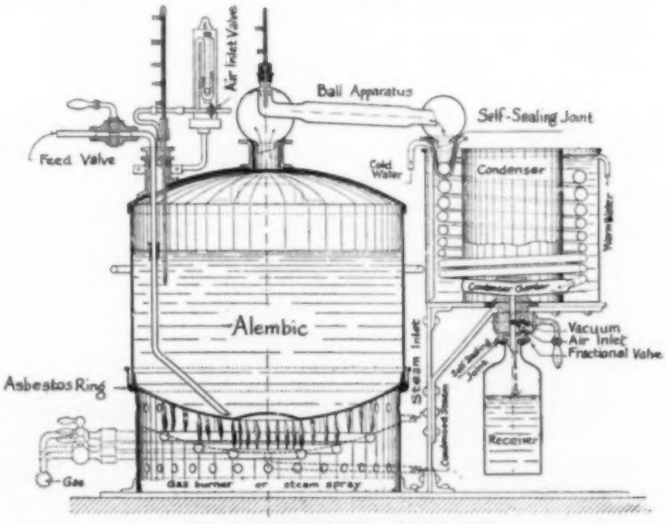
Fractional Distillation Apparatus.

There has recently been introduced in France a new type of distillation apparatus by J. Robert, for all kinds of distillation processes either under atmospheric pressure or in vacuum. It is described in the July, 1911, issue of the *Bulletin de la Société d' Encouragement pour l'Industrie Nationale.*

The arrangement of the parts and the principles of construction and operation may be seen from inspection of the sectional drawing.

The apparatus consists essentially of the alembic, the collector and the condenser. The alembic is a vertical copper boiler with special heads and is equipped for heating either by gas or steam. The bottom head is provided with a concave depression in the center having the same diameter as the center burner; and in this way when the liquid is very low in the boiler it will be prevented from settling in the center and allowing that part of the boiler which is directly over the fire to become overheated.

The top head has two openings, one for the insertion of the inlet pipe, the thermometer and the tube leading to the pressure manometer, and the other leading to the condenser. The



FRACTIONAL DISTILLATION APPARATUS.

neck of the outlet connection is ground slightly conical and is fitted with a leather or rubber ring into which sets a glass ball which forms the collection chamber for the vapor. This form of joint automatically tightens itself as long as the pressure inside the vessel is lower than that on the outside. The vapor which collects in this ball is taken from the top by an inverted tube which is so arranged as to allow all condensate in the ball to drain back into the main vessel. The other end of the outlet tube is joined to the condenser by a similar self-sealing ball joint as may be seen in the illustration.

The condenser consists of a spiral coil immersed in running water. The pipe is large in diameter at the beginning and tapers uniformly to a smaller diameter at the opposite end where it terminates in a chamber at the bottom of the condenser.

The valve which closes the outlet of the condenser is one of the features of the apparatus. It leads directly to the vessel in which the condensate is collected. The joint between the valve and the vessel is made by a metal and a rubber ring which are tightly pressed between the lower end of the valve and the neck of the bottle by the action of the external air pressure, thus automatically sealing the joint.

Whenever it is desired to empty the contents of the vessel the valve handle is turned to the horizontal position, where the passages to the condenser and the vacuum pump are closed and air is admitted to the vessel thus allowing its removal

In replacing the vessel, it is held against the packing rings and the valve handle turned 45 deg., in which position the air inlet is closed and the vessel is connected to the vacuum pump. When the vacuum is established the valve is turned to the vertical position which places it again in communication with the condenser.

It only requires from 4 to 6 seconds to change the vessel and it is done without disturbing in the least the operation of the still.

The advantages claimed for this new type of distilling apparatus are numerous. It is simple and durable in construction and can be operated by unskilled labor. It is easily taken apart and cleaned and is especially well adapted for safe transportation. In operation it is extremely convenient. It may be operated continuously without interruption for an indefinite period. Samples of the distillate can be taken at any time and new liquid can be introduced whenever required without in any way disturbing the vacuum. These characteristics enable the apparatus to be operated without overheating and with a minimum waste of material and time. It is equally well adapted to industrial processes and laboratory analyses.

The principal dimensions and output of several of these stills are given in the following table:

DIMENSIONS AND OUTPUT OF ROBERT STILLS.

Volume of alembic in liters.	Weight of substance to be distilled, grams.	Output per hour, kilograms. Fractional distillation.	Simple distillation.	Total height of alembic, mm.	Diameter of alembic, mm.
17	10—11	4—6	10—12	360	300
68	45—55	10—15	30—35	540	480
136	100—110	17—22	50—60	750	570

Notes.

A large crusher shaft recently repaired by the Goldschmidt thermit process indicates the application of that method to large pieces of mining and metallurgical machinery. The central shaft of a gyratory crusher belonging to the North River Stone Company, Kingston, N. Y., was 18 ft. long and varied in diameter from 12 to 22 in. At the point of fracture the shaft was 14 in. diameter. A wax mold surrounded by a sheet-metal box, packed with ground fire brick and clay, was built around the fracture, leaving a space of 2½ in. between the ends to be welded. Eleven hundred pounds of thermit, with which was mixed 25 per cent of small steel punchings, 1 per cent of chromium thermit and 1 lb. of pure carbonless manganese, was used in making the weld. After making the weld the shaft was allowed to cool for 40 hours before being inspected.

The Philadelphia Coppersmithing Co., Philadelphia, Pa., will be in their new shop, 222-224 North Front St., by Jan. 1. It is said that their new plant is one of the most modern coppersmithing shops in the country. Part of its equipment consists of seven traveling cranes. The Philadelphia Coppersmithing Company can now handle the very largest copper work. The American Evaporator Company will have offices in common with them. The apparatus sold by the American Evaporator Company will be made in this shop.

The Electro-Flex Steel Company of Dunston-on-Tyne, England, are installing an electric-steel furnace of the Stassano type.

The Reading Steel Casting Co. and the Bayonne Steel Casting Co. have been consolidated under the name of the Reading-Bayonne Steel Casting Company. The combined capacity of the two plants at Reading, Pa., and Bayonne, N. J., is from 800 tons to 1000 tons per month. They are equipped for the manufacture of medium and light-weight Trogeans steel castings.

The Florence-Goldfield mill, recently destroyed by fire, probably will be rebuilt as soon as insurance adjustment can be made. At the time of the fire the mill was running only one shift, while waiting for further mine development.

A battery of Macquisten tubes, 24 in number, has been installed at the property of the Grasselli Chemical Company, Park City, Utah, for the re-treatment of old tailings containing zinc and lead. The flotation system is expected to be successful in making a recovery where ordinary wet methods have failed.

The Richards Pulsator Classifier has been designed in a new form, known as the launder type, to meet the demand for classifiers which will furnish a classified product for one table or for any number, by simply attaching it to the launder at a suitable location near each table. The new design is expected to fill a need which the standard types do not meet.

The Colorado Grubstake and Mining Company has been organized as a permanent development of the popular Grubstake movement which originated last summer. The new company is a welcome example of the application of "safe and sane" promotion, and it remains to be seen how it will be received by the public. It has the backing of the Denver Chamber of Commerce, most of the officers of the company being officers and directors of the Chamber. Thomas Tonge, 408 Boston Bldg., Denver, Colo., is acting as secretary of the new company. The results of the prospecting which was carried on last year will be taken over by the company, as also will any promising developed or undeveloped mining properties which can be purchased on terms that would be acceptable to any of the large private exploration companies.

Personal.

Mr. Philip Argall, manager of Stratton's Independence, Ltd., has returned from London, where he went to attend the annual meeting of his company.

Mr. F. M. de Beers, general manager of the Swenson Evaporator Company, of Chicago, has gone on a business trip to Europe.

Mr. H. H. Claudet, who has until recently represented the Elmore Concentration Company in Rossland, B. C., is superintendent of the Wakefield concentrator of the Silverton mines.

Mr. Albert Ladd Colby, consulting engineer, of New York City, has returned from a three months' study of the latest developments in German By-product Coke Manufacture.

Mr. George E. Collins, of Denver, Col., has been elected president of the Colorado Scientific Society for 1912.

Dr. James Douglas has returned to New York from London. While abroad Dr. Douglas delivered an illustrated lecture on "The Development of the Copper Queen and Warren Mining District," before the Institution of Mining & Metallurgy.

Mr. D. L. H. Forbes has opened offices in Toronto, Ontario, where he will represent the Merrill Metallurgical Company as a consulting metallurgist.

Mr. Justin H. Haynes, consulting metallurgist of Denver, Col., who recently completed his work at the Hudson mill, Idaho Springs, Col., has gone to Ophir, Col., where he will be engaged for some months in experimental work at the Suffolk property.

Dr. Wilbur A. Hendryx is now in Johannesburg, South Africa, installing one of his large agitators. It is said that Dr. Hendryx has met with great success in his work in South Africa.

Mr. D. F. Sheehan has been appointed general manager of the Yampa Smelting Company, Utah, to succeed Mr. J. W. D. Moodie, resigned.

Mr. Edward S. Wiard, consulting engineer, of Denver, Colo., is on an extended tour East in the interest of clients.

Mr. Philip D. Wilson has severed his connection with the Copper Queen Mining Company, Douglas, Ariz., and has gone to Cumpas, Sonora, Mexico, for the Transvaal Copper Mining Company.

Digest of Electrochemical United States Patents Prior to 1903.

Arranged according to subject matter and in chronological order.

Compiled by Byrnes, Townsend & Brickenstein, Patent Lawyers, National Union Building, Washington, D. C.

ORE TREATMENT (*Continued*).

507,130, Oct. 24, 1893, Carl Hoepfner, of Berlin, Germany. Copper and silver. Dissolves cupric chloride in a saturated solution of sodium, calcium or magnesium chloride, and uses the solution to leach ore, matte or pyrites. The copper, silver and lead are readily dissolved, especially in sulfids. If the charge contains copper oxids, an acid or other solvent is added. The dissolved silver may be first precipitated, by copper or copper sulfid, with agitation, or by electrolysis, or both the silver and copper may be simultaneously electrodeposited, the solution being passed through a cell having separate anode and cathode compartments arranged to form a zigzag path. The silver will be deposited at the first cathodes; the copper on succeeding cathodes. The electrolyte at the anodes is converted from the cuprous to the cupric form, and is afterward mixed with the solution flowing from the cathodes, to reform the leaching solution. The anodes are carbon and the cathodes copper. The diaphragms are parchment paper reinforced on one or both sides with cloth, leather, veneer, gelatine, asbestos, etc. As the solution generally contains iron, lead, arsenic, antimony, bismuth and cobalt, these may also be electrodeposited with the copper and silver, as alloys. It is preferred, however, to first precipitate them out, as by calcium carbonate. The electrolyte is also treated with calcium carbonate to remove the excess of iron. Or, air or oxygen may be injected into the electrolyte, producing oxy-chlorid of copper which percipitates the iron. The injected air, plus hydrochloric acid, may be used to convert the cuprous chlorid into the cupric form. Sulfurous acid may also be used. Caustic alkalies, alkaline-earths, carbonates, etc., may be used as precipitants for the impurities.

512,361, Jan. 9, 1894, Parker C. Choate, of New York, N. Y., Assignor to The Electrical Zinc Company, of New Jersey.

Zinc. Treats complex ores containing, besides zinc, gold, silver, iron, lead, arsenic, antimony, bismuth, cadmium, etc. The ore is first crushed and roasted at a moderate temperature to drive off any excess of sulfur. It is then mixed with coal screenings and roasted on a grate with air, at a temperature high enough to volatilize the zinc, antimony, arsenic, bismuth and cadmium; also lead. Or the charge may be subjected to a preliminary muffle roast at a temperature of from 500 to 800° F., to drive off the impurities other than lead. The iron, gold, silver, copper, nickel, etc., remain in the furnace in the form of cinder or slag, which may afterward be smelted for their recovery. The volatilized fume is caught in bags, and is electrolyzed to remove first the impurities and then the zinc. The current-density used for purifying is preferably greater than that used to deposit the zinc. Some of the impurities deposit at the cathode and others precipitate; to prevent their re-solution, the free acid liberated by the current is neutralized by the addition of more fume; or it may be drawn off as it rises to the top, neutralized in a separate tank and returned. Enough sulfuric acid should be introduced into the bath to deposit any lead. The clear solution, freed from impurities, is electrolyzed, with a carbon anode and zinc or copper cathodes, yielding commercially pure zinc.

512,362, Jan. 9, 1894, Parker C. Choate, of New York, N. Y., Assignor to The Electrical Zinc Company, of New Jersey.

Zinc. Leaches roasted ore with dilute sulfuric acid. The solution contains, besides zinc, iron, aluminium, manganese, magnesium, copper, nickel, bismuth, cadmium. The insoluble constituents, lead, gangue, etc., are settled out, and the solution is drawn off and heated to evaporate the water and crystallize the contained salts, which are recovered mostly in the form of hydrated sulfates. These are charged into a muffle furnace and heated in an oxidizing atmosphere at from 500 to 900° F., with agitation, thereby eliminating water and constituents more volatile than zinc, and decomposing sulfates other than zinc. The residual anhydrous zinc hydrate, containing also iron, copper and nickel oxids, is removed and treated with water. The zinc sulfate dissolves and is electrolyzed. The sediment is smelted.

No. 513,324, Jan. 23, 1894, H. Hirsching, assignor of one-half to Thomas R. Ellerbeck.

Precious metals. Precipitates the metals in sodium or calcium hyposulfite solutions as sulfids, by electrolysis. The cell is a vessel of wood saturated with paraffin, with suspended plate electrodes of carbon or metal. These electrodes act as agitators, being carried by a revolving frame. Contact is made by circular rings having grooves filled with mercury, into which dip the fixed terminals. The solution is preferably heated to 140° F. The complete precipitation is ascertained by use of hydrogen sulfide solution. The residual solution is again used for leaching ore, a small amount of hyposulfite being added. The precipitated sulfids are easily reduced. Impure silver may be refined by treating it with nitric acid and sodium chlorid to form silver chlorid, and mixing the latter with the sodium hyposulfite solution.

No. 518,543, April 17, 1894, H. F. Edwards, assignor of five-sixths to C. H. Hand, A. W. Hand, J. G. Merrill, G. H. Smith and S. Merrill.

Gold and silver. An electrolytic amalgamator, consisting of a vessel of wood having in the bottom a cathode layer of quicksilver, or amalgamated plates. The anodes are carbon plates secured to the sides of the vessel. Crushed ore mixed with a solution of potassium cyanide is charged into the vessel and is agitated by a revolving stirrer during the treatment. Some of the gold and silver dissolves and is re-precipitated. The free particles of gold and silver drop into the mercury.

BOOK REVIEWS.

The Principles of Electric Ware Telegraphy and Telephony. By J. A. Fleming. 906 pages, 432 illustrations. Price $7.50. New York: Longmans, Green & Co.

The subject matter of this volume represents an elaboration of lectures on radiotelegraphy delivered by Professor Fleming before various British societies and universities during the past ten years. In this, the second edition of the book, which was first published in 1906, much has been inserted on radiotelephony and the treatment of radiotelegraphy has been revised.

The author has covered in an instructive manner the phenomena involved in wireless communication, and has described in sufficient detail the various types of devices that have proved successful for this purpose. After discussing the production of high-frequency currents and electric oscillations, high-frequency electrical measurements, damping and resonance, the author deals with stationary electric waves on wires and electro-magnetic waves and describes numerous equipments for the detection and measurement of electric waves. Chapters are devoted to the evolution of electric wave telegraphy, the appliances of electric wave telegraphy, radiotelegraphic stations and radiotelephony.

No attempt has been made to avoid mathematics; in fact, mathematical treatments have been freely employed. However, in each case the treatment is as simple as the relations involved will permit to be used.